MATERIALS SURVEY ON COPPER

Prepared for the National Security Resources Board

By the United States Department of the Interior, Bureau of Mines

CONTENTS

FIGURES

SUMMARY

CHAPTER II

CHAPTER III

CHAPTER IV

CHAPTER V

CHAPTER VI

STRUCTURE OF INDUSTRY IN FOREIGN COUNTRIES

CHAPTER IV

CHAPTER V

TEXT TABLES

SUMMARY

CHAPTER I

CHAPTER II

CHAPTER III

CHAPTER IV

SUMMARY

1. Relative Importance

Copper is generally regarded as the most important of the nonferrous metals. In tonnage and value of ore produced in the United States, copper is surpassed only by iron. Production of the five leading metals in the United States in 1950 and in the peak year of World War II was as follows:

	1950 production (short tons)	Peak World War II production (short tons)	1950 production as percentage of World War II high
Steel	96,836,000	89,642,000 (1944)	108
Copper	1,500,000	1,529,000 (1942)	98
Lead	991,000	968,000 (1941)	102
Zinc	910,000	991,000 (1943)	92
Aluminum	962,000	1,234,000 (1943)	80

The extensive industrial use of copper depends chiefly on its: (1) Electrical conductivity, (2) corrosion resistance, (3) ductility, and (4) heat conductivity. The commercial importance of copper is also closely related to its alloying properties. It is widely used alloyed with zinc to form brass and alloyed with tin to form bronze. Copper forms many other important alloys both as a base and as a minor constituent when united with numerous other metals, of which beryllium, aluminum, nickel, silicon, and lead are outstanding.

The principal use of copper is in the electrical and allied industries for transmission lines, other forms of conductors, and machinery. The automobile and building-construction industries are the second- and third-largest consumers of copper in the United States during peacetime.

Copper is, moreover, extremely valuable in wartime. Its importance during World War II is indicated by the fact that about 800 pounds of the metal were required for a tank, 1 ton for a large bomber, and 1,000 tons for a battleship. Huge quantities were required for ammunition. A 37-mm. antiaircraft gun used 1 ton of copper every 20 minutes that it was in action, and a 50-plane squadron expended approximately 7 tons of copper in 1 minute of action. Huge tonnages of copper were also needed for more conventional uses during wartime, principally in communications.

2. United States Supply-Distribution

Supply and distribution data may be analyzed by several methods, depending on the purpose for which they are to be used.

In this report, supply is indicated as the total refined copper produced for domestic and foreign ores, refined imports, and unalloyed secondary production. Distribution consists of domestic consumption and exports. Additions to the national stockpile during and after World War II are not included in consumption totals. The following table gives the salient statistics of the copper industry for four periods of the 1925-50 period: 1927-29; pre-World War II (1936-38); World War II (1942-44); and post-World War II (1948-50). Figure 1 shows copper supply and distribution in the United States for 1925-50.

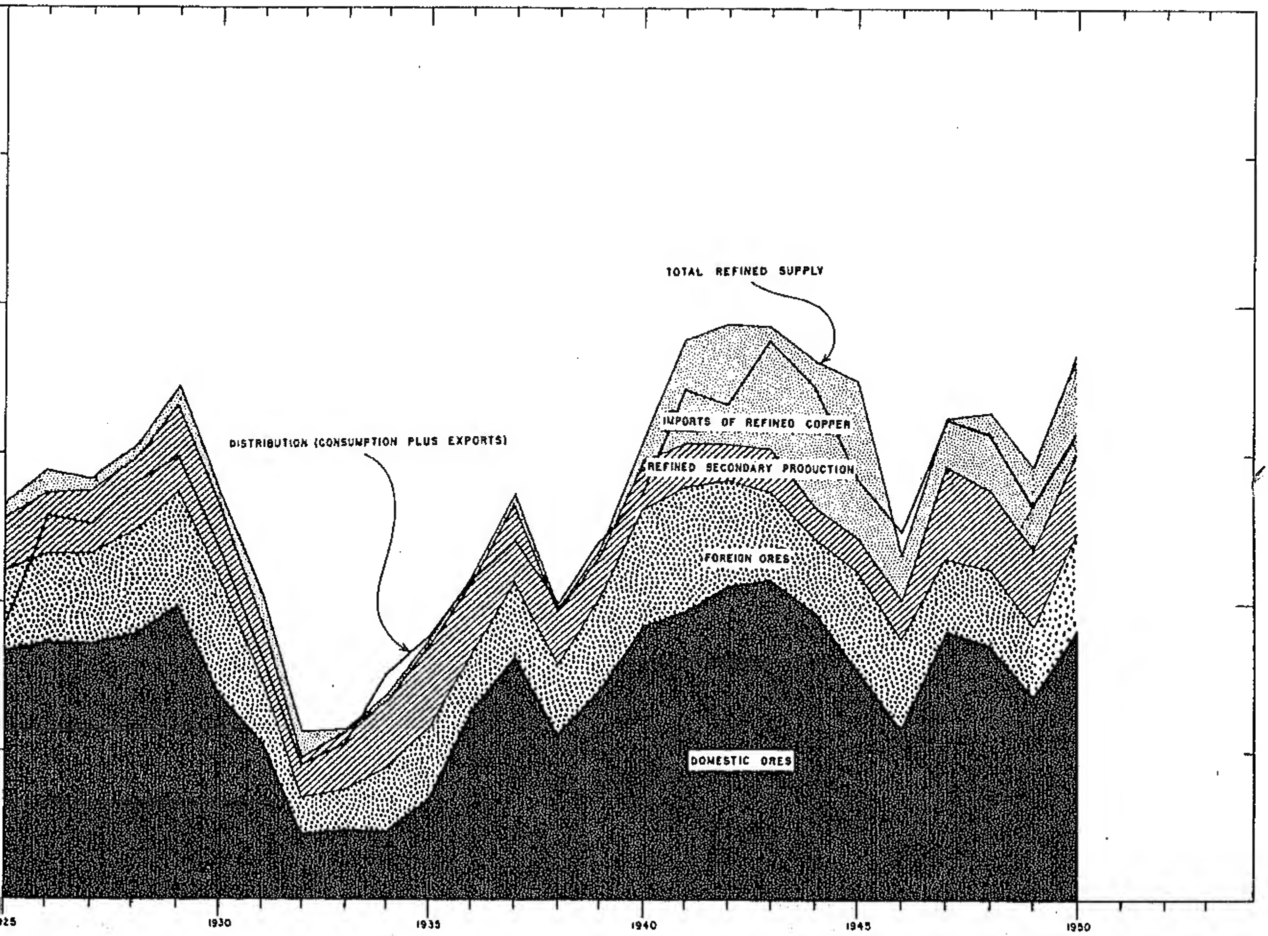


Figure S-1 Copper supply and distribution in the United States, 1925–1950

SALIENT STATISTICS OF THE COPPER INDUSTRY, 1927-29, 1936-38, 1942-44, 1948-50
(Quantities in thousands of short tons)

	1927-29 average	1936-38 average	1942-44 average	1948-50 average
Refined supply: Total	1,555	1,145	1,889	1,637
From domestic ores	915	673	1,040	825
From foreign ores	343	221	298	266
Refined secondary	243	246	118	265
Refined imports	54	5	432	281
Distribution: Total	1,387	1,091	1,770	1,485
Refined domestic consumption	938	796	1,646	1,343
Refined exports	449	295	125	142
Mine production of recoverable copper	909	672	1,048	832
Stocks of refined copper at primary refineries (average of end of year stocks)	72	155	78	51
Average price of electrolytic copper quoted at New York, f.o.b. refinery (cents per pound)	15.32	10.98	11.87	21.01

INDEXES OF ACTIVITY IN THE COPPER INDUSTRY, 1927-29, 1936-38, 1942-44, 1948-50
(1925-29 = 100)

	1927-29 average	1936-38 average	1942-44 average	1948-50 average
Refined supply: Total	104	77	127	110
From domestic ores	103	76	117	93
From foreign ores	108	70	94	84
Refined secondary	109	111	53	119
Refined imports	92	8	732	476
Distribution: Total	109	81	132	116
Refined domestic consumption	105	89	185	151
Refined exports	117	65	28	37
Mine production	103	76	118	94
Stocks of refined copper at primary refineries	90	194	97	64
Price, electrolytic copper	103	74	80	142

3. United States Supply

The United States has been the largest copper producer and consumer in the world throughout the 1925-50 period. Production of refined copper from domestic ores for the decade 1941-50 averaged 884,000 tons a year. Mine production and secondary recovery were adequate to meet all domestic requirements and to provide significant quantities for export before World War II. When the United States entered World War II, however, large additional supplies of copper were required from abroad, and the United States became a net importing nation. This situation has continued through 1950.

The total supply of copper to the United States reflected, to a great extent, the economic conditions of the country throughout the 26 years 1925-50. The total refined supply, from domestic and foreign ores, refined secondary copper, and refined imports, averaged about 1.5 million tons in 1925-29, dropped sharply during the depression of the thirties, increased markedly with the advent of war in Europe in 1939, and rose to nearly 2 million tons annually during the peak years of World War II. In periods of comparable prosperity, 1925-29 and 1946-50, however, the total supply of copper was approximately equal -- 1,490,000 and 1,539,000 tons, respectively.

While total supply did not change substantially during comparable periods of the 26-year period, the quantities derived from various sources of supply changed considerably.

Mine production: Mine production of copper changed relatively little over the period 1925-50. The average yearly output was 832,000 tons in 1948-50, compared with 909,000 tons per year in 1927-29. The greatest 3-year total production during the 26-year period, and the all-time high was attained in 1942-44 when the annual output averaged 1,048,000 tons. Premium payments and subsidies for overquota and marginal production contributed to the expanded output during World War II. The increased production was obtained principally from the Bingham, Utah, Morenci, Ariz., Butte, Mont., Central, N. Mex., and Ray, Ariz., districts.

The bulk of new copper in the United States comes from open-pit operations. Mass mining has made possible the working of extremely low-grade ores. This essentially nonselective technique has brought about increased efficiency of labor. Labor productivity, in terms of pounds of recoverable copper per man-day, has more than doubled over the 1925-50 period, despite a sharp decrease in the grade of ore mined.

Refinery production: The production of refined copper depends upon domestic mine output, imports of ores and concentrates, and scrap recovery. Refinery production from these sources has exceeded 1,000,000 tons a year every year since 1939 (exception 1946), reaching a peak of 1,502,000 tons in 1943. In 1948-50 the average yearly output was 1,308,000 tons and was obtained from 14 plants, 10 of which employed the electrolytic method, 2 the furnace process on Lake Superior copper, and 2 the furnace process on western ores.

In 1950 primary refining capacity totaled approximately 1,827,000 tons, which, on the basis of 1948-50 production, represented 72 percent utilization.

Secondary production: Secondary copper is produced from new and old scrap, the two general classes of scrap copper. Old scrap consists of metal articles that have been discarded because of wear, damage, or obsolescence, usually after serving a useful purpose. New scrap is defined as refuse accumulated during the manufacture of articles for ultimate consumption, and is generally considered as "run-around copper", that is, not adding to the total supply. Secondary copper is classified by the form in which recovered-alloyed and unalloyed- and only the unalloyed class is considered to be part of the total new supply of refined copper. Only copper from old scrap, whether in unalloyed or alloyed form, is a real addition to supply.

The use cycle of copper is considered to be about 40 years, and it is estimated that about 60 percent of the copper put into use is eventually recovered.1/ Old scrap copper therefore, provides a large and readily accessible source of the metal.

The production of secondary copper has increased tremendously since the beginning of World War II. Recovery, including both alloyed and unalloyed copper, averaged 888,000 tons a year in 1948-50, 61 percent above the average yearly output in 1936-38. The highest recorded production - 1,086,000 tons - was in 1943. Since 1941 over three-fourths of the total secondary output has been recovered in alloys, principally brass and bronze. During the 10 years 1941-50, production of unalloyed copper from scrap has accounted for less than one-quarter of the total scrap recovery. In 1948-50 unalloyed production averaged 265,000 tons a year or 16 percent of the total refined supply of copper.

Imports: One of the most significant developments in the copper industry during 1925-50 has been the increased dependence of the United States on imports to maintain adequate supplies.

1/ Skelton, Alex, International Control in Nonferrous Metals, Macmillan Co., New York, 1937, pp. 384, 385.

This country was a net exporter of copper until 1940-41, when defense requirements exceeded the domestic supply by a wide margin. Imports of unmanufactured copper (including copper content of ores and concentrates, regulus, and unrefined copper, refined, and old scrap) increased from a yearly average of 241,000 tons in 1936-38 to 491,000 tons in 1940 and 853,000 tons in 1945, the all-time high. It should be emphasized, moreover, that imports were more than balanced by exports in the pre-World War II period, while in the 1940's exports comprised only a small portion of total distribution. During World War II, 1941-45, refined copper produced from foreign ores and concentrates and copper imported in refined form together accounted for approximately 40 percent of the total refined copper supply of the United States. Imports as refined metal comprised nearly 60 percent of the total supply from foreign sources. In the postwar period, 1946-50, approximately 33 percent of the total refined-copper supply of the United States was of foreign origin.

United States imports have come largely from properties owned by American investors. Chile, Mexico, and Peru, countries whose copper industries are dominated by United States companies, and Canada, have consistently been the largest sources of imports, annually supplying about three-fourths of the total for the 1925-50 period.

United States imports from Chile during and since World War II were several times as large as before the war. This noteworthy increase represents primarily a diversion of shipments of Chilean copper from Europe to the United States. Before the war, Chile exported less than one-fifth of its copper output to the United States; since 1946 that nation has shipped about half of its annual output to this country. Large quantities of copper were obtained from the Belgian Congo during World War II, when that country was cut off from its European markets.

4. United States Distribution

The distribution of copper in the United States (domestic consumption and exports) increased substantially in 1925-50; average yearly distribution was 1,485,000 tons in 1948-50 compared to 1,387,000 tons in 1927-29. Distribution reached an all-time peak of 1,892,000 tons in 1944 and averaged 1,757,000 tons yearly during the war years, 1941-44.

The distribution pattern changed greatly over the 26-year period; domestic demands consumed an increasingly larger percentage of the total, while exports declined in importance. In 1948-50 distribution was composed of domestic consumption (90 percent) and exports (10 percent), whereas in 1927-29 consumption comprised 68 percent and exports 32 percent, respectively, of the total.

Consumption: Domestic utilization of copper rose markedly from 1925 through 1950. Consumption totaled 1,461,000 tons a year for the 10 years 1941-50, 64 percent above the 1925-29 average annual usage of 891,000 tons. Consumption reached an all-time high of 1,718,000 tons in 1943 and averaged 1,586,000 tons a year in 1941-45. Although copper was in a "critical commodity" classification throughout World War II, supply and distribution were carefully controlled, and no serious shortages developed for military and essential civilian uses.

The predominant use of copper is for electrical purposes; approximately 50 percent of the yearly consumption in this country is used by the electrical industries. The automotive and building industries are other important users of copper, each consuming about 10 percent of the annual total. Wire mills and brass mills generally account for more than 95 percent of total consumption with wire mills taking the larger share in peacetime and brass mills greater tonnages during wartime.

Exports: Copper exports have diminished in both quantity and percentage of the total distribution since the beginning of World War II. Exports of refined copper declined from an average of 449,000 tons per year or 32 percent of total distribution in 1927-29 to 142,000 tons or less than 10 percent of distribution in 1948-50. The chief importers of United States copper during recent years were the United Kingdom, France, India, the Netherlands, and Italy. Currently much of the refined copper exported is copper that has been refined on toll.

5. World Supply-Distribution Position

The copper production-consumption position for the world's leading producing and consuming nations in 1950 is shown by the following table.

WORLD PRODUCTION AND CONSUMPTION OF COPPER IN 1950
(Quantities in thousands of short tons)

Country	Mine production Quant.	Mine production % of total	Smelter production Quant.	Smelter production % of total	Consumption Quant.	Consumption % of total
World total	2,749		2,961		2,980	
Canada	262	9.5	240	8.1	105	3.5
Cuba	23	.8	-	-	-	-
Mexico	68	2.5	53	1.8	6	.2
United States	909	33.1	1,008	34.0	1,372	46.0
Brazil	-	-	-	-	23	.8
Chile	397	14.4	380	12.8	21	.7
Peru	33	1.2	25	.8	-	-
Belgium	-	-	-	-	64	2.1
Cyprus	26	.9	-	-	-	-
Finland	17	.6	15	.5	9	.3
France	-	-	-	-	127	4.3
Germany	1	-	221	7.5	238	8.0
Italy	-	-	-	-	64	2.1
Sweden	18	.7	18	.6	54	1.8
United Kingdom	-	-	-	-	374	12.6
U.S.S.R.	240	8.7	240	8.1	243	8.2
Yugoslavia	44	1.6	44	1.5	33	1.1
India	8	.3	7	.2	35	1.2
Japan	43	1.6	93	3.1	115	3.9
Turkey	15	.5	13	.4	a/	-
Australia	16	.5	15	.5	20	.7
Belgian Congo	194	7.1	194	6.6	a/	
Northern Rhodesia	328	11.9	309	10.4	a/	
Union of South Africa	50	1.8	37	1.2	a/	
Others	57	2.1	49	1.7	77	2.6

a/ Data not available; not large.

The United States is the world's greatest copper producer, annually furnishing about one-third of the total world output. Chile, with mine output of nearly 20 percent of the world total during the 1940's, is the second-largest supplier of copper. Northern Rhodesia, Canada, the U.S.S.R., and the Belgian Congo are other outstanding copper producers. Together, these six countries supply 80 to 85 percent of the world's annual output of copper (mine and smelter production).

The United States is also by far the largest user of copper; in 1950 this nation accounted for almost half of the entire world consumption. The United Kingdom ranks second in utilization of the metal, followed by the U.S.S.R., Germany, France, Japan, and Canada. These seven countries were responsible for more than 85 percent of the total world consumption in 1950. The United States, the United Kingdom, France, and to a large extent, Germany and Japan, require large quantities of imports to fulfill domestic requirements. Chile, Northern Rhodesia, the Belgian Congo, and Canada are the principal exporters of copper.

World refining capacity totaled approximately 4,390,000 tons in 1950, with the United States possessing the greatest facilities - about 1,827,000 tons annually. On the basis of a smelter production of 2,961,000 tons (world-wide refinery production statistics are not available) in 1950, output represented 67-percent utilization of capacity.

The world production-consumption position during several periods of the 1925-50 interval is given in the following table.

WORLD PRODUCTION AND DISTRIBUTION OF COPPER
1927-29, 1936-38, 1942-44, 1948-50
(Quantities in thousands of short tons)

	1927-29 average		
	Mine production	Smelter production	Consumption
World total	1,910	1,884	1,951 a/
North America	1,103	1,125	996
South America	384	351	b/
Europe	167	175	814
Asia	91	82	84
Australia	12	12	10
Africa	153	139	4

	1936-38 average		
	Mine production	Smelter production	Consumption
World total	2,249	2,244	2,223 a/
North America	990	991	791
South America	418	398	10 c/
Europe	299	368	1,182
Asia	122	109	197
Australia	21	18	16
Africa	399	359	5

	1942-44 average		
	Mine production	Smelter production	Consumption
World total	2,918	2,977	b/
North America	1,401	1,491	
South America	593	569	
Europe	284	277	
Asia	134	143	
Australia	27	24	
Africa	479	473	

	1948-50 average		
	Mine production	Smelter production	Consumption
World total	2,594	2,749	2,783 a/
North America	1,174	1,214	1,360
South America	467	432	45
Europe	365	497	1,185
Asia	66	106	149
Australia	14	13	21
Africa	508	487	18

a/ Includes small quantities undistributed.
b/ Data not available.
c/ Estimated.

World mine production of copper increased substantially over the 1925-50 period; the average yearly output in 1948-50 was approximately 2,600,000 tons, almost 700,000 tons more than the 1927-29 average. During World War II, with copper in very short supply owing to the extraordinary demand for the metal, mine production averaged better than 2,900,000 tons per year. World production declined considerably immediately after the war, increased in 1947, and by 1950 was near the World War II rate. Africa made the most significant gains during the 26-year period owing to Northern Rhodesia's emergence as a major copper producer.

Smelter production followed the same general pattern as mine production from 1925 through 1950. World output exceeded 3 million tons per year in 1942-43, over 75 percent higher than the 1925-29 average yearly output. Production declined during the postwar period, increased beginning in 1947, and in 1950 was slightly under 3 million tons. It is of interest to note the rapid recovery of the European industry following World War II. The average yearly output in 1948-50 surpassed the prewar rate of production, largely because of increased ore production in the U.S.S.R. and greater scrap recovery in Germany. Production from the other continents decreased after World War II.

Figures 2 and 3 show world mine and smelter output of copper by countries for 1950. The predominance of the six largest producers is evident.

WORLD PRODUCTION AND DISTRIBUTION OF COPPER
1927-29, 1936-38, 1942-44, 1948-50
(Quantities in thousands of short tons)

1927-29 average	Mine production	Smelter production	Consumption
World total	1,910	1,884	1,951 a/
North America	1,103	1,125	996
South America	384	351	b/
Europe	167	175	814
Asia	91	82	84
Australia	12	12	10
Africa	153	139	4

1936-38 average	Mine production	Smelter production	Consumption
World total	2,249	2,244	2,223 a/
North America	990	991	791
South America	418	398	10 c/
Europe	299	368	1,182
Asia	122	109	197
Australia	21	18	16
Africa	399	359	5

1942-44 average	Mine production	Smelter production	Consumption
World total	2,918	2,977	b/
North America	1,401	1,491	
South America	593	569	
Europe	284	277	
Asia	134	143	
Australia	27	24	
Africa	479	473	

1948-50 average	Mine production	Smelter production	Consumption
World total	2,594	2,749	2,783 a/
North America	1,174	1,214	1,360
South America	467	432	45
Europe	365	497	1,185
Asia	66	106	149
Australia	14	13	21
Africa	508	487	18

a/ Includes small quantities undistributed.
b/ Data not available.
c/ Estimated.

World mine production of copper increased substantially over the 1925-50 period; the average yearly output in 1948-50 was approximately 2,600,000 tons, almost 700,000 tons more than the 1927-29 average. During World War II, with copper in very short supply owing to the extraordinary demand for the metal, mine production averaged better than 2,900,000 tons per year. World production declined considerably immediately after the war, increased in 1947, and by 1950 was near the World War II rate. Africa made the most significant gains during the 26-year period owing to Northern Rhodesia's emergence as a major copper producer.

Smelter production followed the same general pattern as mine production from 1925 through 1950. World output exceeded 3 million tons per year in 1942-43, over 75 percent higher than the 1925-29 average yearly output. Production declined during the postwar period, increased beginning in 1947, and in 1950 was slightly under 3 million tons. It is of interest to note the rapid recovery of the European industry following World War II. The average yearly output in 1948-50 surpassed the prewar rate of production, largely because of increased ore production in the U.S.S.R. and greater scrap recovery in Germany. Production from the other continents decreased after World War II.

Figures 2 and 3 show world mine and smelter output of copper by countries for 1950. The predominance of the six largest producers is evident.

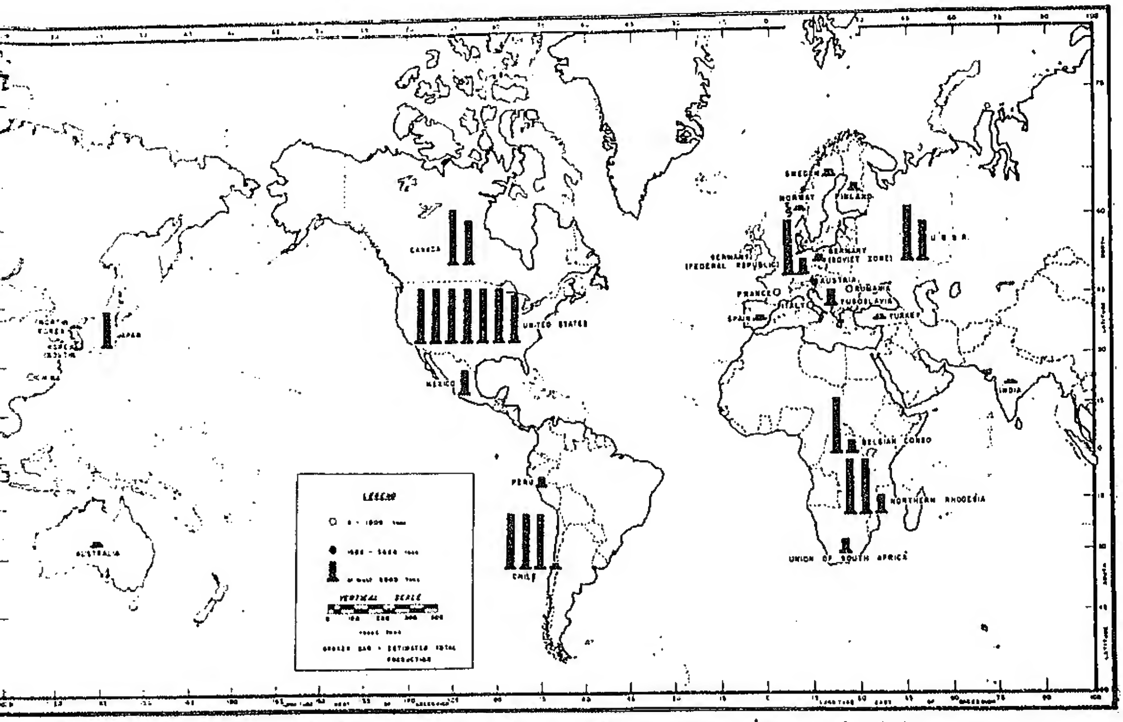


Figure S-2 World smelter production of copper by country, 1949, in short tons

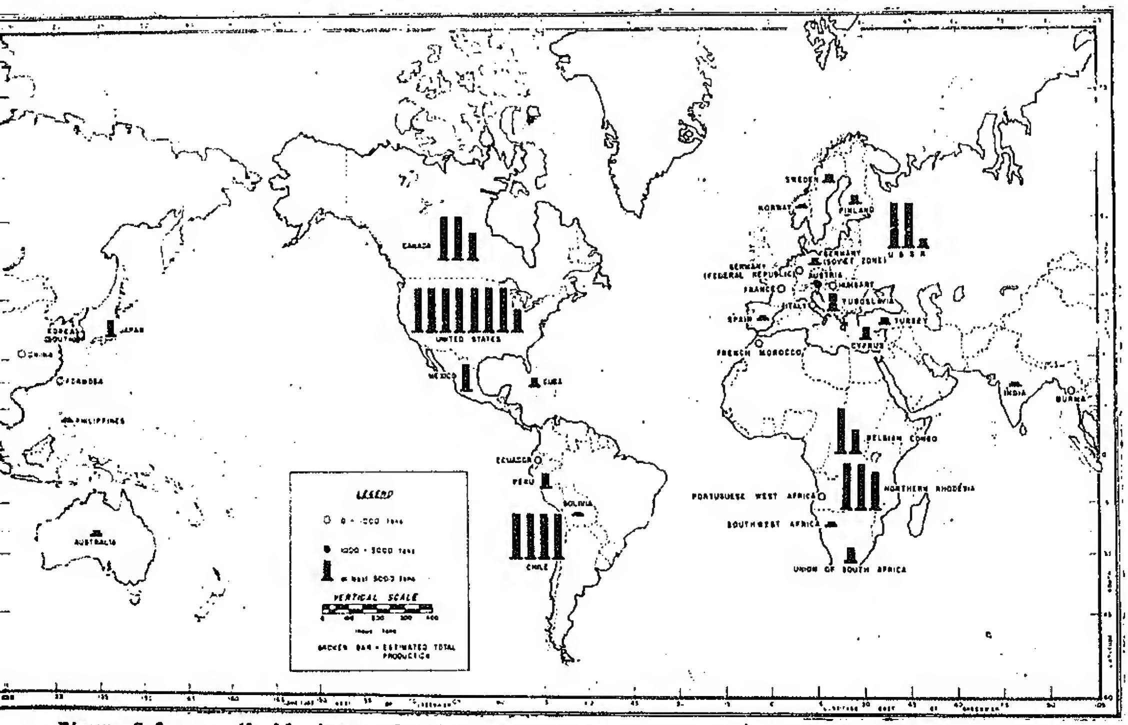


Figure S-3 World mine production of copper by country, 1949, in short tons

6. Resources

Copper is so widespread in nature that almost every country has some copper-ore deposits; 18 countries produced over 10,000 tons of recoverable copper in 1950, and about 18 other nations recorded some output. Yet the major part of the world's known copper is concentrated in a few places. The scrap supply is chiefly in the industrial areas, and about 90 percent of the world's unmined copper resources is located in four regions - south-central Africa, Chile, the western United States, and Kazakhstan, U.S.S.R., in that order. In Canada the Sudbury, Ontario, district and southern Quebec may be comparable to Kazakhstan as a copper region, but comprehensive data on reserves are lacking. The following table lists 12 districts or mines containing 85 percent of the world's copper resources. This list is a compromise between developed reserves that are surely economic under present conditions and partly explored semieconomic deposits that are so large that they probably will be important for the future. Deposits that are not known to contain copper reserves in quantities greater than 3,000,000 tons of copper metal have been omitted from the list.

TWELVE DISTRICTS OR MINES CONTAINING 85 PERCENT
OF THE WORLD'S COPPER RESOURCES,
JANUARY 1, 1950

Deposits	Country	Major ownership	Nationality
1. "Mine Series"	No. Rhodesia	Selection Trust, Ltd., Anglo-American Corp., Ltd.	British
2. Chuquicamata	Chile	Anaconda Copper Mining Co.	U. S.
3. "Mine Series"	Belg. Congo	Union Minière du Haut Katanga	Belgian
4. Butte, Mont.	U. S.	Anaconda Copper Mining Co.	U. S.
5. Braden (El Teniente)	Chile	Kennecott Copper Corp.	Do.
6. Bingham, Utah	U. S.	Do.	Do.
7. Keweenaw, Mich. a/	Do.	Copper Range Co. & Calumet & Hecla Consolidated Copper Co.	Do.
8. Morenci, Ariz.	Do.	Phelps Dodge Corp.	Do.
9. Sudbury, Canada b/	Canada	International Nickel Co., of Canada, Ltd., and Falconbridge Nickel Mines, Ltd.	Canadian
10. San Manuel, Ariz. a/	U. S.	Magma Copper Co. Newmont Mining Co.	U. S.
11. Kazakhstan	U.S.S.R.	State-owned	U.S.S.R.
12. Urals region	Do.	Do.	Do.

a/ Large reserves are considered marginal awaiting production tests.
b/ Copper is a byproduct of nickel production.

Coverage is raised to 93 percent by the addition of 13 districts, as follows:

United States - Ely-Kimberly, Nev.; Ray, Ariz.; Chino, N. Mex.; Ajo, Ariz.; Bisbee, Ariz.; Yerington, Nev.; Miami-Inspiration, Ariz.
Mexico - Cananea
Chile - Potrerillos (Andes Copper Co.)
Aguirre - Africana (Santiago), Rio Blanco
Peru - Toquepala - Quelleveco, Cerro de Pasco

The remaining 7 percent of the world's reserves is distributed among 34 countries.

Ultimate copper resources are an important aspect for long-range planning, but reserves are normally considered in terms of economically profitable material. The United States reserve position, in the light of current profitability, ranks first, 2/ at present, among the three major copper regions of the world, but the United States may be a poor third in the long run. The main reason is the relatively lean quality of domestic ore.

The African deposits are notable for huge reserves of 3- to 6-percent ore, whereas the United States average is less than 1 percent. The Chilean deposits have huge tonnages averaging 2 percent copper. U.S.S.R. grades appear to be no better than those of the United States.

A physical factor favoring the United States competitive position is that three-fourths of our production now comes from large-scale, low-cost, open-pit deposits in contrast to the underground mines in South America 3/ and Africa. However, American open-pit deposits face eventual termination because of increasingly adverse ratios of waste rock that must be removed to uncover copper ore. Under current conditions, at least, the lives of the American open-pit operations are numbered - some will last for 10 years, some 20 years, and those that persist another 30 years will be exceptions. Some, however, may be able to change to underground mining.

Recent discoveries and major unequipped properties in the United States are low-grade deposits. On the basis of the physical

---

2/ On the basis of apparent costs of production cited by Malozemoff, P. - The Real Danger of High Costs: Eng. and Min. Jour., vol. 150, Jan. 1950, p. 72.
3/ The great exception is Chuquicamata, which will operate an open pit for many years more while underground operations are developed.

racteristics of the deposits, it must be emphasized that most rican deposits have a temporary present advantage but a distinct g-range disadvantage compared with the richer, larger African Chilean deposits.

The United States is in a healthy position with respect to the ply of copper known to remain in the ground. The present major lucing mines have 20 to 30 years of life at the production rates 1940-50. Beyond this, the exploration campaigns of the past ade around old mines and in new areas indicate an additional 10 .5 years supply. If intensive exploration is continued, it is ected that this favorable position will be maintained. An impor- feature of the copper reserve picture is that copper is a large- e, mass-production resource. About 95 percent of the reserves r in less than a dozen mines or mining areas. These deposits, uding both economic and semi-economic material are listed in the owing table.

TWELVE MINING DISTRICTS CONTAINING 95 PERCENT
OF THE UNITED STATES COPPER RESERVES, JANUARY 1, 1950

District	Principal ownership
Butte, Mont.	1. Anaconda Copper Mining Co.
Bingham, Utah	2. Kennecott Copper Corp.
Keweenaw, Mich. a/	3. Copper Range Co. & Calumet & Hecla Consolidated Copper Co.
Morenci, Ariz.	4. Phelps Dodge Corp.
San Manuel, Ariz. a/	5. Magma Copper Co. (Newmont Mining Co.)
Ely-Kimberly, Nev.	6. Kennecott Copper Corp. Consolidated Coppermines Corp.
hino, N. Mex.	7. Kennecott Copper Corp.
ay, Ariz.	8. Do.
jo, Ariz.	9. Phelps Dodge Corp.
erington, Nev. a/	10. Anaconda Copper Mining Co.
iami, Ariz. b/	11. Miami Copper Co. Inspiration Consolidated Copper Co.
isbee, Ariz.	12. Phelps Dodge Corp.

ow being equipped for production.
ncludes Castle Dome, Copper Cities (Sleeping Beauty), ration, Miami, and Globe.

Addition of the following mines and districts brings the coverage to 98 percent: Magma, Ariz.; Silver Bell, Ariz.; Cornwall, Pa.; Bagdad, Ariz.; Tyrone, N. Mex.; and Glacier Peak, Wash. The remaining 2 percent of reserves is distributed among some 200 present and former copper mines and districts of the United States.

The critical factor in utilizing copper resources under normal conditions is the highly competitive market. The economic conditions for profit and loss may change rapidly, but a huge quantity of copper is available in the ground. For each fraction of a percen that the grade of ore is lowered, large tonnages of copper are addec to the supply.

7. Structure of Industry in the United States

The primary copper industry of the United States is composed of approximately 200 firms engaged in the production and sale of copper. The principal segments of the industry - mining, smelting, refining, fabricating, and marketing - are dominated in varying degrees by a few large vertically integrated companies.

Mining: Three companies, Kennecott Copper Corp., Phelps Dodge Corp., and Anaconda Copper Mining Co., generally produce over three-fourths of the annual domestic mine production of copper. Kennecott and Phelps Dodge are by far the largest producers, supplying about 40 percent and 30 percent of the total, respectively, each year. The principal producing companies, with 1950 production, are given in the following table.

PRINCIPAL COPPER-PRODUCING COMPANIES IN THE UNITED STATES, 1950

Company	Mine production, short tons	Percent of total
Kennecott Copper Corp.	418,000	46.0
Phelps Dodge Corp.	245,000	26.9
Anaconda Copper Mining Co.	50,000	5.5
Inspiration Consolidated Copper Co. (Anaconda holds 28 percent of the issued stock)	38,000	4.2
Miami Copper Co. (including Castle Dome Copper Co., Inc.)	46,000	5.1
Calumet & Hecla Consolidated Copper Co.	22,000	2.4
Magma Copper Co.	24,000	2.6
Tennessee Copper Co.	a/	
Total	843,000	92.7
Total all companies	909,000	

a/ Data not available.

There were approximately 300 active copper-producing mines in the United States in 1950. Most of them were small operations, however, for the 25 largest mines produced 98 percent of the total domestic output. The top 5 mines produced 67 percent of the total, and the 10 leading mines furnished 85 percent of the aggregate. The following table lists the 10 leading mines in order of 1950 output.

Mine	District	State	Operator	Type of ore
Utah Copper	West Mountain (Bingham)	Utah	Kennecott Copper Corp.	Copper ore
Morenci	Copper Mountain (Morenci)	Arizona	Phelps Dodge Corp.	Do.
New Cornelia	Ajo	Do.	Do.	Do.
Chino	Central	N. Mex.	Kennecott Copper Corp.	Copper, zinc-lead ore
Butte Mines	Summit Valley	Montana	Anaconda Copper Mining Co.	Do.
Ruth Pit	Robinson (Ely)	Nevada	Kennecott Copper Corp.	Copper ore
Inspiration	Globe-Miami	Arizona	Inspiration Consolidated Copper Co.	Copper, zinc-lead ore
Ray	Mineral Creek (Ray)	Do.	Kennecott Copper Corp.	Copper ore
Miami	Globe-Miami	Do.	Miami Copper Co.	Do.
Castle Dome	Globe-Miami	Do.	Castle Dome Copper Co., Inc.	Do.

Copper-producing areas are located principally in the Western States. Following is a list of the more important districts, with 1950 production.

District	State	1950 production, short tons
West Mountain (Bingham)	Utah	278,000
Copper Mountain (Morenci)	Arizona	155,000
Globe-Miami	Do.	85,000
Ajo	Do.	64,000
Central (including Santa Rita)	N. Mex.	64,000
Summit Valley (Butte)	Montana	54,000
Robinson (Ely)	Nevada	52,000
Mineral Creek (Ray)	Arizona	36,000
Lake Superior	Michigan	26,000
Pioneer (Superior)	Arizona	23,000
Warren (Bisbee)	Do.	13,000
Verde (Jerome)	Do.	13,000
Eureka (Bagdad)	Do.	11,000

These districts produced 96 percent of the 1950 mine output (909,000 tons). The bulk of the remainder was obtained from smaller producing districts in the Western States and from the Southeastern Missouri, Tennessee, Lebanon, Pa., and Orange County, Vt., districts.

Smelting: Copper-smelting capacity in the United States in 1950 totaled approximately 9,748,000 tons of charge. Four companies control about 93 percent of this capacity. These companies, with the other copper-smelting companies, and their approximate percentage ownership of the total are listed below, in order of magnitude of available facilities.

Company	Percent of total capacity
Phelps Dodge Corp.	41.5
American Smelting & Refining Co.	29.6
Anaconda Copper Mining Co.	14.0
Kennecott Copper Corp.	7.6
Magma Copper Co.	2.6
American Metal Co., Ltd.	2.1
Calumet & Hecla Consolidated Copper Co.	1.0
Copper Range Co.	.9
Tennessee Copper Co.	.7
Quincy Mining Co.	.1

Because copper mining is centralized principally in the Western States, the greater part of the smelting capacity is in that area. There is some capacity in Michigan to take care of the Michigan mines, some capacity on the east coast to take care of Eastern production and imports, and one smelter in Tennessee.

Refining: Copper-refining capacity in the United States in 1950 aggregrated about 1,827,000 tons, composed of electrolytic capacity of 1,599,000 tons and fire-refinery capacity of 228,000 tons. Three companies dominate this section of the copper industry, controlling over three-fourths of the annual capacity.

These companies, along with the other refining companies, are listed below with percent ownership of capacity in 1950.

Company	Percent of total capacity
American Smelting & Refining Co.	26.6
Phelps Dodge Corp.	23.5
Anaconda Copper Co.	21.3
Kennecott Copper Corp.	11.8
American Metal Co., Ltd.	8.8
Calumet & Hecla Consolidated Copper Co.	2.7
Copper Range Co.	2.7
Inspiration Consolidated Copper Co.	2.1
Quincy Mining Co.	.3

ne greater part of the United States refining capacity is
l on the Atlantic seaboard in metropolitan New York and New
and Baltimore. Cheap power, so important in electrolytic
ng, and large near-by markets together with ocean transpor-
have combined to produce this concentration. Electrolytic
ries are also located at El Paso, Tex., Great Falls, Mont.,
spiration, Ariz., where low-cost electric power is also
ole.

abrication: Fabricators are the principal customers of the
copper producers. It is in the fabricating plants that the
f the new copper is put into semifinished forms - wire, rods,
ed and rolled shapes, etc. - which constitute the raw materials
ny other industries.

bout 30 companies in the United States are generally recog-
as important fabricators and users of raw copper, the latter
for the most part, the large electrical manufacturers such
eral Electric, Westinghouse, etc. The more important of the
ators, representing over 50 percent of the total volume of
ss, are owned or controlled by the great copper producers,
them completely integrated operations from the mines to the
ed brass and copper products. A list of the fabricating com-
controlled by these companies follows.

FABRICATING COMPANIES OF PRINCIPAL COPPER PRODUCERS

Fabricating Company	Controlled by
Brass & Copper Co. ott Wire & Cable Co.	Kennecott Copper Corp. Do.
n Brass Co. a Wire & Cable Co.	Anaconda Copper Mining Co. Do.
Dodge Copper Products Division n Tube Works, Inc. aw Cable & Wire Corp.	Phelps Dodge Corp. Do. Do.
Copper & Brass Co., Inc. Cable Corp.	American Smelting & Refining Co. (owns 36.45 percent of stock) (owns 24.53 percent of stock)
ne Tube Corp.	Calumet & Hecla Consolidated Copper Co.
ussey & Co.	Copper Range Co.
en Copper Co.	Tennessee Corp. (parent company of the Tennessee Copper Co.)

The more important independent fabricators not now controlled by the major producers include the following:

Bridgeport Brass Co.
Scovill Manufacturing Co.
Okonite Co. (wire and cable)
Mueller Brass Co.

Secondary Production: The recovery of scrap copper constitutes an important branch of the copper industry in the United States. Old scrap is largely collected by several hundred scrap dealers, who, for the most part, are unaffiliated with the great primary producers. Notable exceptions include the Federated Metals Division of the American Smelting & Refining Co., perhaps the largest scrap-copper dealer in the country, the American Metal Co., Ltd., and the Anaconda Sales Co., a subsidiary of the Anaconda Mining Co.

The channels through which much of the reclaimed copper returns from the scrap dealers and fabricators to use are the secondary refiners, the brass and bronze ingot makers, and the brass mills.

The bulk of the secondary copper in this country is recovered in ingot brass and bronze and other alloys; recovery of scrap as unalloyed copper represents less than one-third of the total annual output.

Marketing: The market for raw copper in the United States is confined to a very limited number of buyers, probably not exceeding 60, including the fabricators integrated with the large primary producers. The transactions of about 30 of these purchasers represent the bulk of all raw copper sales. The principal users are the fabricators affiliated with the large producers, the independent fabricators, and the large electrical manufacturers. The large producing companies channel their output of refined copper to their fabricating plants and eventually sell the bulk of it not as a commodity but in finished products. The independent fabricators and the electrical manufacturers, buying on the open market, often experience great difficulty in securing the copper they require.

Copper is usually sold on a basis of 90-day deliveries from the refineries, the copper producers handling their transactions with the consumers through their sales agents, which, in the case of the large producers, are generally subsidiaries or affiliated companies. The American Metal Co., Ltd., and the American Smelting & Refining Co., both custom smelters and refiners, are the principal independent selling organizations. Adolph Lewisohn & Sons, Inc., is also an important seller of copper, acting as sales agent for the Tennessee Copper Co. and the Miami Copper Co. (including Castle Dome Mining Co. and Copper Cities Mining Co.). Calumet &

Hecla Consolidated Copper Co., Magma Copper Sales Corp., Consolidated Coppermines Corp., and International Minerals & Metals Corp. are other notable primary copper sellers.

The large integrated producers maintain price leadership in the copper industry, setting prices in the short run to a marked extent. Their major consideration is a satisfactory operating profit over a long period of years and not necessarily a definite margin for each sale. Unlike the custom smelters, the integrated producers might refrain from selling in a falling market rather than lower prices. The principal custom smelters, American Smelting & Refining Co. and American Metal Co., Ltd., also play a prominent role in setting prices. These companies sell refined metal against ore intake in order to hedge against possible losses, and in a weak market this practice speeds drops. There is no community of interest between these two selling groups. Supply-demand factors, of course, are of great importance in long-run price determination.

Copper prices are expressed in cents per pound determined by the market activity in New York. Prices are generally quoted in terms of electrolytic copper, f.o.b. refinery, with quotations for the other types of copper fluctuating around the electrolytic price.

8. The Copper Industry During World War II

Copper was in critically short supply throughout all of World War II. Controls were maintained over producers and consumers of copper during this period to insure adequate supplies for military purposes and essential civilian uses.

The measures taken to balance supply and distribution included increasing imports greatly, raising and maintaining mine production at a high level, increasing secondary output, decreasing exports, reducing nonessential consumption, and increasing the usage of less critical materials as substitutes for copper.

The need for conservation of copper became evident soon after the outbreak of war in Europe in 1939. Consumption of refined copper in the United States increased from 828,000 tons in 1939 to 1,013,000 tons in 1940 and 1,614,000 tons in 1941. Domestic mine production and the recovery of scrap were unable to meet the expanded demand. Large additional supplies were therefore required.

Imports: One of the first actions taken was to increase imports. Imports made available as refined copper (refined imports and refined copper produced from ores, concentrates, and unrefined material) averaged 765,000 tons per year, 41 percent of our total refined supply, during 1941-45. The pre-World War II, 1935-39, average was 253,000 tons, 23 percent of the total refined supply.

The bulk of United States imports was obtained from Chile during World War II. Mexico, Canada, and Peru also supplied large quantities of copper to this country during the war period, as did Belgian Congo, when its normal European markets were cut off by German occupation of the continent. Losses of water-borne imports by sinkings were nominal.

The Government purchased virtually all of the foreign copper entering the country throughout the war period, making the 4-cent-per-pound tax on imports ineffective.

Mine Production: The extraordinary consumption of copper during World War II made it imperative that domestic mines produce at their maximum rate. Several measures were taken to assure a steady, high output. One of the most important of these methods was the Premium Price Plan, inaugurated in February 1942. This plan was instituted primarily to increase domestic mine production by paying premium prices for production above a certain quota, which was based upon 1941 output. Through payment of these premiums, many marginal and submarginal mines were operated, and tailings and slag piles, previously unprofitable to treat, were reworked. This plan and other practices, notably the employment of more men in production of ore at the expense of development work, resulted in a substantial rise in mine production. Output increased from 728,000 tons in 1939 to 1,091,000 tons in 1943, averaging 975,000 tons per year for 1941-45, 56 percent greater than the 1935-39 average. From the peak of 1943 mine output declined, largely because of a serious manpower shortage brought about by a general migration of labor away from the mines to higher-paying industries and the drafting of miners.

During the time the Premium Price Plan was in effect, from February 1942 through June 1947, 22 percent of all the copper produced domestically received premiums. The average price paid per pound was 14.28 cents, less than 2 cents above the ceiling.

Additional measures taken to increase mine production included the exploration and drilling programs of the Geological Survey and the Bureau of Mines of the U. S. Department of the Interior. Several potential producing areas were brought to attention through these programs, adding substantial tonnages to the domestic ore reserves.

Distribution Controls: The demand for copper exceeded the available supply throughout most of the World War II period, necessitating administration of regulatory measures over distribution.

Copper was placed under mandatory industry-wide controls, effective July 1941. The order provided for setting aside a producers'

pool to be allocated by the Director of Priorities and filling of defense orders according to preference ratings before any shipments could be made for civilian use.

The Office of Production Management established a priorities system of allocating copper. Limitation and conservation orders were issued to reduce civilian consumption, prohibit its use for nonessential purposes, and effect conservation of copper. This system was not very successful principally because it did not provide for integration of the production of component parts with over-all programs such as tanks, planes, and ships.

Early in 1943 the War Production Board installed the Controlled Materials Plan, entailing a new system for controlling critical materials. With respect to copper, the purpose of this plan was to assure a balance between the supply and demand of copper so that the metal would be available in sufficient quantity at the proper time to meet all authorized program schedules. This plan remained in effect through the remainder of the war period.

Stockpiling: One of the initial steps taken by the Government to assure stability of prices and to prevent development of shortages in copper was to recommend in November 1940 that the Reconstruction Finance Corporation, through the Metals Reserve Company, purchase foreign copper.

The Metals Reserve Company maintained a stockpile of copper from 1942 through 1947. The following table indicates the magnitude of these stocks.

GOVERNMENT COPPER STOCKS AT END OF YEAR, 1942-47
(In short tons)

1942	91,472
1943	224,081
1944	412,635
1945	565,710
1946	92,758
1947	9,986

No copper remained with the Metals Reserve Company at the end of 1948, existing stocks having been sold to industry or transferred to the National Strategic Stockpile.

The Government stockpile served a vital purpose; it was available to emergency war use. It was largely dissipated, however, in the postwar reconversion period, the largest withdrawals occurring in strike-ridden 1946.

I. PROPERTIES, PRODUCTS, AND USES

A. HISTORICAL SKETCH

The early history of copper is one of the most interesting chapters in human development, for its use marked the epochal advance made by man from the Stone Age to the Bronze Age. The Bronze Age was a stage of civilization, rather than a definite chronological period, through which the peoples of the Near East, Egypt, India, China, and Central America passed at varying times. The utilization of copper preceded the Bronze Age by at least a thousand years, but it was not until the hardening qualities of an additional tenth part of tin were discovered that the resulting bronze metal grew to such importance as to christen an era.

The first use of copper is lost in the earliest days of history, for it occured as a native metal in many parts of the world and could be readily used without metallurgical treatment. It probably has been utilized by man for at least 20,000 years. The step from working native copper, which was treated simply as a malleable stone, to smelting copper ores was extremely significant but is of unknown origin. By 3500 B. C. copper ornaments and simple tools and utensils were in common use by the Aegean, Near Eastern, and Egyptian peoples. In the same area bronze was discovered and came into common use by 2200 B. C.

The same fundamentally international basis existed for the copper industry in early times as at present. Bronze articles from Asia Minor were carried up the ancient Danube trade route as far north as Silesia and Saxony. Metallurgical development was stimulated in Bohemia and Saxony and in Spain, and eventually the knowledge of bronze was spread through Europe. From Asia Minor trade routes ran to South Russia, the Near East, and the Western Mediterranean. The Mediterranean traders searched for copper in the period, from Rio Tinto and Cornwall to Russia.

At approximately the same time, parallel developments were taking place in China and India while later, at about the beginning of the Christian era, the stone-copper-bronze cycle occured in Central America. In North America, development had not passed the primitive copper stage, although there is evidence of quite extensive early mining operations in Michigan.

The breakdown of the slave economy following the decline of the Roman Empire curbed copper mining until new forms of power - gunpowder and steam - arose. In the Middle Ages production again began to expand in the old metallurgical centers - Bohemia and Saxony and the Iberian Peninsula.

In the early development of copper mining, operations were generally confined to relatively rich ores and to those near the

surface. Other factors as well tended to prevent large production, but by the end of the eighteenth century greatly increased ability to produce copper had resulted from the invention of gunpowder, used in blasting rock; the mine pump that freed many abandoned workings of water, thus permitting a resumption of mining operations; and, finally, the steam engine, used for hoisting. Better understanding of smelting had made reduction of ores more efficient and less costly, and laws that had formerly given the bulk of metal production to the sovereign or the landowner had been amended liberally.

It is therefore probable that the annual world production of about 18,000 tons of copper at the beginning of the nineteenth century was in itself the culmination of a great increase in production that had been in progress for several centuries.

World copper production increased phenomenally from 1800 through 1949, particularly during the last 75 years. The development of the United States copper industry is given in Chapter VI, Section A, and the growth of the world industry since 1800 is covered in Section D of that chapter.

B. THE METAL: ITS ALLOYS AND COMPOUNDS

1. Copper Metal

Copper ranks next to iron as a metal of commercial importance. It is important primarily because it is available in abundance, has the best conductivity of any base metal, and useful alloys it forms. The following is a table of some important physical and chemical properties of elemental copper.

PHYSICAL PROPERTIES OF COPPER

Chemical symbol	Cu
Atomic number	29
Atomic weight	63.54
Density at 68° F., gm./cc.	8.96
lb./cu. in.	0.321
Melting point, °F.	1981.4
Boiling point, °F.	4700
Coefficient of linear-thermal expansion near 68° F., micro-in./°F.	9.2
Electrical resistivity, microhm-cm.	1.673
Crystal structure	Face-centered cubic
Valence	1 & 2

Copper is particularly interesting in the electrical field since it is the material of which transmission wires and the other forms of electrical conductors, including parts of dynamos, motors, switch boards, etc., are usually made. It has the highest electrical conductivity of any metal or substance except silver. The electrical conductivity of copper is 94 percent that of silver, while that of the next highest metal, gold, is only 66 percent that of silver and iron has only 16 percent of the conductivity of silver. Copper has enough strength for minor structural purposes (such as sheet-metal work, electrical manufactures, etc.), is easily rolled and drawn into wire, has great resistance to weathering, and is of moderate cost compared with competitive materials. In addition to these properties, copper is widely used alloyed with zinc to form brass, which is easily worked, offers good resistance to weathering and most solutions (principal exceptions are certain acids and alkalies), and is fairly strong and elastic; and alloyed with tin to form bronze, of note for its high high resilience. It has good thermal conductivity, so finds many uses in heat-transfer units, such as cooling fins and water heaters. In addition, a large percentage of copper may be recovered as scrap after it has outlived the usefulness for which it was originally intended. Of the total copper consumed in the United States it has been estimated that about 60 percent eventually returns to use as copper or copper alloys.

Commercial Classes

There are several types of commercial copper, graded principally according to their chemical composition and electrical conductivity. Accepted universally by the trade are the specifications of the American Society for Testing Materials.(1) 1/ Specifications have been adopted for the following types of copper:

1. Lake Copper Wire Bars, Cakes, Slabs, Billets, Ingots, and Ingot Bars (must originate on the northern peninsula of Michigan).
   (a) Low Resistance Lake Copper - minimum purity of 99.900 percent, silver being counted as copper.
   (b) High Resistance Lake Copper - minimum purity of 99.900 percent, silver and arsenic being counted as copper.
2. Electrolytic Copper Wire Bars, Cakes, Slabs, Billets, Ingots and Ingot Bars - minimum purity of 99.900 percent, silver being counted as copper.
3. Electrolytic Cathode Copper - minimum purity of 99.90 percent, silver being counted as copper.
4. Electrolytic Copper, Oxygen-Free, Wire Bars, Billets, and Cakes - minimum purity of 99.92 percent, silver being counted as copper.
5. Fire-Refined Casting Copper - must conform to the following requirements as to chemical composition:

	Percent
Copper plus silver, minimum	99.70
Arsenic, maximum	.100
Antimony, maximum	.012
Bismuth, maximum	.003
Iron, maximum	.010
Lead, maximum	.010
Nickel, maximum	.100
Oxygen, maximum	.075
Selenium, maximum	.040
Tellurium, maximum	.014
Tin, maximum	.050

6. Fire-Refined Copper For Wrought Products and Alloys - must conform to the following requirements as to chemical composition:

	Percent
Copper plus silver, minimum	99.88
Arsenic, maximum	.012

1/ Numerals in parenthesis refer to items in the selected references section of the bibliography at the end of the chapter.

Antimony, maximum	.003
Selenium plus tellurium, maximum	.025
Nickel, maximum	.05
Bismuth, maximum	.003
Lead, maximum	.004

The American Society for Testing Materials has established a tentative classification of coppers. This classification, with definitions, is shown on the following table.

CLASSIFICATION OF COPPERS

Desig-nation	Type of copper a/	Forms in which available b/ — From refiners c/: Wire bars	Billets	Cakes	Ingots and ingot bars	From fabricators d/: Flat products	Pipe and tube	Rod and wire	Shapes
CATH	Electrolytic cathode	Available in cathodes only							
	Tough Pitch Coppers								
ETP	Electrolytic tough pitch e/	X	X	X	X	X	X	X	X
FRHC	Fire-refined high-conductivity tough pitch e/	X	X	X	X	X	X	X	X
FRTP	Fire-refined tough pitch	...	X	X	X	X	X	...	X
ATP	Arsenical, tough pitch	...	X	X	...	X	X	X	...
STP	Silver-bearing tough pitch e/	X	X	X	X	X	X	X	X
SATP	Silver-bearing arsenical tough pitch	...	X	X	...	X	X	...	...
SETP	Selenium-bearing tough pitch	...	...	...	...	...	...	X	X
TETP	Tellurium-bearing tough pitch	...	...	...	...	...	...	...	X
CAST	Casting	...	...	...	X	...	...	...	...
	Oxygen-Free Coppers								
OF	Oxygen-free without residual deoxidants e/	X	X	X	f/	X	X	X	X
OFS	Oxygen-free, silver-bearing e/	X	X	X	...	X	...	X	X
OFTE	Oxygen-free, tellurium-bearing	...	...	...	...	X	...	X	X
	Deoxidized Coppers								
DHP	Phosphorized, high residual phosphorus	X	X	X	...	X	X	X	X
DLP	Phosphorized, low residual phosphorus g/	...	X	X	...	X	X	...	X
DPS	Phosphorized, silver bearing	...	X	X	...	X	X	...	X
DPA	Phosphorized, arsenical	...	X	...	...	X	X	...	...
DPTE	Phosphorized, tellurium bearing	...	...	...	...	...	...	X	X

a/ See Section 3 and Appendix 1.
b/ The X in the table indicates commercial availability.
c/ See Appendix II.
d/ See Appendix III.
e/ Types ETP, FRHC, STP, OF, AND OFS are high-conductivity coppers.
f/ Croppings of other shapes available and can be used as ingots.
g/ Type DLP can be furnished as a high-conductivity copper if agreed upon between the supplier and the purchaser.

APPENDIX I

DEFINITIONS OF TERMS USED IN CLASSIFICATION OF COPPERS

Copper:-For the purpose of this classification, copper containing less than 0.5 percent of alloying elements has been included in the term "copper."

Terms Relating to Method of Refining

Electrolytic Copper.-Copper which has been refined by electrolytic deposition, including cathodes which are the direct product of the refining operation, refinery shapes cast from melted cathodes, and, by extension, fabricators' products made therefrom. Usually when this term is used alone, it refers to electrolytic tough pitch copper without elements other than oxygen being present in significant amounts.

Fire-Refined Copper.-Copper which has been refined by the use of a furnace process only, including refinery shapes, and, by extension, fabricators' products made therefrom. Usually when this term is used alone, it refers to fire-refined tough pitch copper without elements other than oxygen being present in significant amounts.

Terms Relating to Characteristics Determined by Method of Casting or Processing

Tough Pitch Copper.-Copper either electrolytically or fire refined, cast in the form of refinery shapes, containing a controlled amount of oxygen for the purpose of obtaining a level set in the casting. By extension, the term is also applicable to fabricators' products made therefrom.

Oxygen-Free Copper.-Electrolytic copper, free from cuprous oxide produced without the use of residual metallic or metalloidal deoxidizers. By extension, the term is also applicable to fabricators' products made therefrom.

Deoxidized Copper.-Copper cast in the form of refinery shapes, free from cuprous oxide through the use of metallic or metalloidal deoxidizers. By extension, the term is also applicable to fabricators' products made therefrom.

Terms Relating to Specific Kinds of Coppers

High-Conductivity Copper.-Copper which, in the annealed condition, has a minimum electrical conductivity of 100 percent I.A.C.S. as determined in accordance with A.S.T.M. methods of test.

Casting Copper.-Fire refined tough pitch copper usually cast from melted secondary metal into ingot and ingot bars only, and used for making foundry castings, but not wrought products.

Phosphorized Copper.-General term applied to copper deoxidized with phosphorus. The most commonly used deoxidized copper.

High Residual Phosphorus Copper.-Deoxidized copper with residual phosphorus present in amounts (usually 0.013 to 0.040 percent) generally sufficient to decrease appreciably the conductivity of the copper.

Low Residual Phosphorus Copper.-Deoxidized copper with residual phosphorus present in amounts (usually 0.004 to 0.012 percent) generally too small to decrease appreciably the conductivity of the copper.

Arsenical Copper, Selenium Bearing Copper, Silver Bearing Copper, Tellurium Bearing Copper.-Copper containing the designated element in amounts as agreed upon between the supplier and the consumer. Any of these alloyed coppers can be produced as tough pitch, oxygen-free, or deoxidized varieties. For the ones commonly supplied see table on page 6.

APPENDIX II

DEFINITIONS OF REFINERY SHAPES

Wire Bar.-Refinery shape for rolling into rod (and subsequent drawing into wire), strip or shape.

> Approximately $3\frac{1}{2}$ to 5 in. square in cross-section, usually from 38 to 54 in. in length and weighing from 135 to 420 lb. Tapered at both ends when used for rolling into rod for subsequent wire drawing and may be unpointed when used for rolling into strip. Cast either horizontally or vertically.

Cake.-Refinery shape for rolling into plate, sheet, strip, or shape.

> Rectangular in cross-section of various sizes. Cast either horizontally or vertically, with range of weights from 140 to 4000 lb. or more.

Billet.-Refinery shape primarily for tube manufacture.

> Circular in cross-section, usually 3 to 10 in. in diameter and in lengths up to 52 in.; weight from 100 to 1500 lb.

Ingot and Ingot Bar.-Refinery shapes employed for alloy production (not fabrication).

> Both used for remelting. Ingots usually weigh from 20 to 35 lb. and ingot bars from 50 to 70 lb. Both usually notched to facilitate breaking into smaller pieces.

Cathode.-Unmelted flat plate produced by electrolytic refining.

> The customary size is about 3 ft. square and about $\frac{1}{2}$ to 7/8 in. thick, weighing up to 280 lb.

APPENDIX III

DEFINITIONS OF FABRICATORS' COPPER PRODUCTS

Wire.-A solid section, other than strip, furnished in coils or on spools, reels, or bucks.

Tube.-A hollow product of round or any other cross-section, having a continuous periphery.

Pipe.-Seamless tube conforming to the particular dimensions commercially known as "Standard Pipe Sizes."

Shape.-A solid section, other than rectangular, square or standard rod and wire sections, furnished in straight lengths.

> Shapes are usually made by extrusion but may also be fabricated by drawing.

Flat Product.-A rectangular or square solid section of relatively great length in proportion to thickness.

> Included in the designation "flat product," depending on the width and thickness, are plate, sheet, strip, and bar. Also included is the product known as "flat wire."

Rod.-A round, hexagonal or octagonal solid section. Round rod for further processing into wire (known as "hot-rolled rod," "wire-rod," or "redraw wire") is furnished coiled. Rod for other uses is furnished in straight lengths.

Effects of Impurities

The conductivity, tensile strength, and other properties of copper are greatly affected by the presence of small amounts of impurities which may (1) be dissolved in the copper in solid solution or (2) be insoluble in solid copper. The most important of those in the first class are nickel, iron, arsenic, antimony, and phosphorus. Those in the second class are bismuth, lead, selenium, tellurium, sulfur, oxygen, and oxides.

Oxygen is present in all commercial copper, except deoxidized grades. Its effect on the mechanical properties is not great, slightly increasing the tensile strength and reducing the ductility as the amount of oxygen increases. In small amounts oxygen increases the electrical conductivity of commercial copper, very likely by oxidizing other impurities and removing these from solid solution. Large amounts of oxygen, however, reduce the conductivity by forming copper oxide, thus reducing the effective cross section of the metal.

Sulfur, selenium, and tellurium, in general, affect the mechanical properties of copper adversely. In amounts up to 1 percent, selenium and tellurium may be used for increasing machinability. Lead, when used for this purpose, causes "hot-shortness".

Bismuth also has a harmful effect on the mechanical properties of copper; it interferes seriously with hot rolling, and somewhat with cold. However, it is rarely present in domestic copper, but is a common impurity in much foreign copper. Its effect may be partly neutralized by additions of oxygen, arsenic, and antimony. It is almost completely insoluble in copper.

On the whole, antimony is another harmful metal, although it is sometimes added to copper where high recrystallization temperatures are desired. (See page I-12.) In amounts of 0.5 percent and higher it hardens copper, decreases its ductility, and lowers its electrical conductivity. Furthermore, antimony is very undesirable in brass.

Arsenic is often added intentionally in amounts up to 0.6 percent by reason of its slight hardening and strengthening effect, especially in the cold-worked condition. It raises the recrystallization temperature but has little effect on the ductility and malleability. It decreases the electrical conductivity considerably.

Next to oxygen, silver is the most common element in commercial copper. It has negligible effect on the mechanical properties and electrical conductivity but has a great effect on the recrystallization temperature.

The small amount of iron normally present in commercial copper has very little effect on its mechanical properties. In larger amounts, up to 2 percent, it hardens and strengthens copper slightly without destroying ductility, but it reduces the electrical conductivity rapidly, especially in the absence of oxygen.

Lead is sometimes added to copper to increase its machinability, but should not exceed 0.005 percent if the copper is to hot-rolled, otherwise it will cause "hot-shortness". If operations are conducted at room temperature, still larger amounts have little effect on the ductility of copper. It can be rendered somewhat less harmful by the introduction of oxygen.

Gases have a great effect on the physical properties of copper, especially in castings. Hydrogen is very soluble in liquid copper but will not cause unsoundness in copper castings, in the absence of oxygen, unless the solubility in the solid state, which is high, is exceeded. Carbon monoxide is probably soluble in solid and liquid copper to about the same extent and in the absence of oxygen is not harmful. Both carbon dioxide and nitrogen behave as if they were insoluble in copper. The influence of gases in copper is a complex problem of great importance to the producers of copper castings. On the whole, the amount of gas evolved in casting should exactly neutralize the natural shrinkage of the metal on passing from the liquid to the solid state.

Heat Treatment

As is true with many other metals and their alloys, the physical properties of copper and the copper alloys may be materially changed by heat treatment. In working copper and copper alloys stresses are introduced by changes in crystal structure, and accompanying hardness and brittleness tend to obliterate desired mechanical attributes. Heating under controlled conditions allows partial recrystallization and relieves stresses, so that the strength, hardness, ductility, etc., are regulated to give the specified qualities to the finished product.

Annealing is the only method of heat treatment used on pure copper, although other heat-treating processes may be used on some copper alloys. The purpose of annealing and other heat-treatment processes is to restore work-hardened copper to its original soft or ductile form.

The process of annealing involves heating the copper to the proper temperature, holding at that level for a certain period, and then allowing the metal to cool to room temperature. Most commercial annealing is done at about 1,100° F., which provides the necessary softening action without permitting undue grain growth.

2. Copper Alloys

Copper fabricators produce a great variety of alloys, which may contain copper, zinc, tin, lead, nickel, iron, phosphorus, arsenic, manganese, silicon, beryllium, and other elements. Many of these are sold under trade names; but most of them are copper-zinc alloys, or brasses, copper-tin alloys or bronzes, or one of a small group of special alloys derived from these to meet specific demands.

Brass

Alpha brass is a solution of zinc in copper containing up to 36 percent zinc. Low brasses containing up to 20 percent zinc are noted for their ability to be worked and formed at room temperature without hardening. High brasses contain 20 to 35 percent zinc, and even though they require a heat-softening process after working are used in nearly every type of fabricated product. Many articles can be heat-treated when fabricated, and whenever this is possible high-zinc brass is used because of the relatively lower cost of zinc compared to copper. A third classification is sometimes made of the high-zinc alloy (60-40 copper zinc) called Muntz metal. The solubility of zinc in copper is exceeded at this composition; as a result, two phases or types of crystals (alpha and beta) are present in the metal. The chief features that distinguish Muntz metal and other mixed alpha-beta brasses from the plain alpha brasses are extreme plasticity at red heat followed by hardening on cooling. This allows die casting and high reductions of thickness on rolling at red heat, yielding a hard, stiff, strong product when cold. The names, compositions, and properties of the principal simple brasses are listed in the following tables.

Good machining characteristics are obtained when lead is added in small amounts to a brass. Lead forms minute globules, which makes the brass chip off instead of spiraling on a lathe tool. The compositions and properties of leaded brasses are given in the following tables.

The tin brasses contain up to 1 percent of tin which is added to brasses to impart good cold-working characteristics, as a coloring agent, and to increase corrosion resistance. The compositions of some of these alloys together with their properties and uses are given in the following tables.

BRASSES BY NAME, COMPOSITION, USES, PROPERTIES, SPECIFICATIONS /a

Alloy name	Gilding, 95%	Commercial bronze, 90%	Red brass, 85% (rich low brass)	Low brass, 80%
Composition:				
Copper	95.0	90.0	85.0	80.0
Zinc	5.0	10.0	15.0	20.0
Lead	-	-	-	-
Typical uses	Jewelry, emblems, plaques, coins, fuse caps, primers, as base for articles to be gold plated or highly polished	Costume jewelry, compacts, lipstick cases, forgings, screws, weatherstripping, stamped hardware	Jewelry, badges, name plates, tags, dials, hardware, etched parts, automobile radiators, tube and pipe for oil and utility fields, and plumbing	Jewelry, thermostat bellows, deep-drawn articles
General properties	Compared to copper, this alloy has higher tensile strength, equal ductility, lower thermal properties, golden color	Very ductile	Higher strength and ductility than copper; excellent corrosion resistance, often exceeding that of copper; more successful at high temperature than higher zinc alloys	Similar to 85 percent Red Brass
Working properties:				
Cold working	Excellent	Excellent	Excellent	Excellent
Hot working	Do.	Do.	Good	Good
Machining	Poor	Poor	Poor	Poor
Welding	Gas, carbon arc, metal arc	Gas, carbon arc, metal arc	/b	/b
Soldering	Excellent	Excellent	Excellent	Excellent
Polishing	Do.	Do.	Do.	Do.
ASTM Specifications:				
Sheet, strip	B36	B36, B130	B36	B36
Rod				
Tube			B43, B111, B135	

BRASSES BY NAME, COMPOSITION, USES, PROPERTIES, SPECIFICATIONS /a

Alloy name	Cartridge brass, 70%	Yellow brass	Muntz metal (Yellow metal)
Composition:			
Copper	70.0	65.0	60.0
Zinc	30.0	35.0	40.0
Lead	-	-	/c
Typical uses	For cartridge cases and ammunition components	For deep drawing, stamping, spinning, etching, rolling for practically all fabricating processes. Pins, rivets, eyelets, auto radiator cores, heating units, lamp bodies, cartridge cases and clips, electrical sockets, etc.	Sheet form for ship sheathing, condenser heads, perforated metal, architectural work, condenser tube, valve stems, brazing rods
General properties	Best combination of ductility and strength of any brass	Excellent cold-working properties combined with good corrosion resistance, mechanical properties and corrosion resistance almost the same as Deep Drawing and Cartridge Brass	High strength combined with low ductility
Working properties:			
Cold working	Excellent	Excellent	Fair
Hot working	Good	Poor	Excellent
Machining	Fair	Fair	Good
Welding	/b	/b	/b
Soldering	Excellent	Excellent	Excellent
Polishing	Do.	Do.	Do.
ASTM specifications:			
Sheet, strip	B19, B36	B36	B171 /c
Rod	B134	B134	
Tube	B135		B111, B135

a/ Source: American Society for Metals, Metals Handbook: 1948 ed.
b/ For welding - gas, carbon arc, metal arc, spot and seam welding for thin gage.
c/ Leaded Muntz metal containing 0.40-0.80 percent lead for improved machinability, is supplied for plate applications.

LEADED BRASS BY NAME, COMPOSITION, USES, PROPERTIES, SPECIFICATIONS /a

Name	Low-Leaded Brass (tube)	High-Leaded Brass (Engravers Brass)	Riveting and turning rod	Free-Cutting Brass	Forging Brass	Architectural Bronze
Composition:						
Copper	67.0	64.0	63.0	61.5	60.0	56.50
Zinc	32.5	34.0	35.25	35.5	38.0	41.25
Lead	0.5	2.0	1.75	3.0	2.0	2.25
Tin						
Typical uses	Screw machine parts, electrical fuse parts, plumbing pipe, pump liners	Engraving plates, machined parts, instruments, professional and scientific name plates, keys, lock parts, tumblers, gears, watch parts	Rivets and fasteners, parts involving machining and slight amount of cold work, such as staking or bending	Deep drilling turning, free cutting for screw machine parts	Hot forgings, hardware, plumbing goods	Handrails, decorative moldings, grills, revolving-door parts, miscellaneous architectural trim, industrial extruded shapes (hinges, lock bodies, automotive parts)
General properties	Free machining, combined with moderate cold-forming ability	Free machining and good blanking	Excellent machining, combined with moderate cold-forming ability	Excellent machinability, combined with good mechanical and corrosion-resistance properties	Extremely plastic hot, combining good corrosion resistance with excellent mechanical properties	Excellent forging and free machining properties
Working properties:						
Cold working	Fair	Poor	Fair	Poor	Fair	Very poor
Hot working	Poor	Do.	Do.	Good	Excellent	Excellent
Machining	Good	Excellent	Excellent	Excellent	Good	Good
Welding	Nonleaded preferred	Nonleaded preferred	Nonleaded preferred	Nonleaded preferred	Nonleaded preferred	Poor
Soldering	Excellent	Excellent	Excellent	Good	Good	Excellent
Polishing	Do.	Do.	Do.	Excellent	Excellent	Do.
ASTM specifications:						
Sheet, strip						
Rod				B16	B124	
Tube	B135					

a/ Source: American Society for Metals, Metals Handbook: 1948 ed.

TIN BRASS BY ALLOY NAME, COMPOSITION, USES, PROPERTIES, SPECIFICATIONS /a

Alloy name	Admiralty	Naval Brass	Leaded Naval Brass	Manganese Bronze	Arsenical Bronze	Aluminum Brass (Special Brass)
Composition:						
Copper	71.00	60.00	60.00	58.50	57.00 Ni 1.8	76.00
Zinc	28.00	39.25	37.50	39.20	38.65 Mn .2	22.00
Lead	As .05		1.75	Fe 1.00	.75 As .6	Al 2.00
Tin	1.00	.75	.75	1.0	Fe 1.0	As .05
				Mn .30		
Typical uses	Condenser and heat-exchange plates and tubes, steam power-plant equipment, chemical and process equipment, marine uses, automobile aerials	Tube sheets in heat exchangers and steam condensers, hot-worked forgings	Screw machine products, machine hardware, forgings, bolts	Forgings, condenser plates, valve stems, coal screens	Valve stems, extruded shapes, forgings	Condenser and heat exchanger tubes, steam power plant equipment, chemical and process equipment, marine uses
General properties	Excellent corrosion resistance, combined with strength and ductility	Resistance to salt-water corrosion, satisfactory for moderate cold-working operations	Similar to Free-Cutting Brass (240) but with increased strength and corrosion resistance	High strength combined with excellent wear resistance	Good wear and corrosion resistance, with excellent machinability	Excellent corrosion resistance combined with good strength and ductility better than Admiralty for resistance to velocity type of failure
Working properties:						
Cold working	Good	Fair	Poor	Poor	Poor	Good
Hot working	Poor	Excellent	Good	Excellent	Excellent	Fair
Machining	Fair	Good	Excellent	Good	Do.	Poor
Welding	Gas, carbon arc	/b	Nonleaded brass preferred	/b		
Soldering	Good	Excellent	Good	Excellent	Excellent	Excellent
Polishing	Do.	Do.	Excellent	Do.	Do.	
ASTM specifications:						
Rod		B21, B124	B21	B124, B136		
Sheet	B171	B171				
Tube	B111					B111

a/ Source: American Society for Metals, Metals Handbook: 1948 ed.
b/ For welding, gas, carbon arc, metal arc, spot and seam welding for thin gage.

225350 O-52-5

Bronze

There are fewer bronzes than brasses. The hardness, strength, and general quality of resilience or springiness increase with tin content; these characteristics, coupled with high fatigue strength (the ability to withstand repeated or oscillating loading), cause bronze to be used extensively for springs.

Most of the modern compositions designed for castings are mixed brass and bronze or brass with some quality, such as color, to justify the term "bronze". Other alloys, such as manganese and aluminum bronzes, are not true bronzes but special alloys. Copper-tin bronzes nearly always contain some deoxidizing agent, usually phosphorus, so many bronzes are commonly designated as phosphor bronzes.

The following tables give the composition, uses, and general properties of some important bronzes.

BRONZE ALLOYS BY ALLOY TYPES, ALLOY NAME, COMPOSITION, USES, PROPERTIES, SPECIFICATIONS /a

Alloy type	Silicon Bronze		Aluminum Bronze	
Alloy name	High-silicon bronze	Low-silicon bronze	Aluminum bronze, 5%	Aluminum silicon bronze
Chemical composition:				
Copper	96.0	98.00	95.0	91.0
Zinc			Al 5.0	Al 7.0
Silicon	3.0	1.50		2.0
Manganese	1.0	.25	As .035	As .035
Tin				
Typical uses	Tanks-pressure vessels steam pressure not to exceed 125 pounds; vats-baskets-marine construction, weatherstrips, forgings, conduits, hydraulic pressure lines	Cold-working-cold headed bolts, nuts, screws, lag bolts-hydraulic pressure line-cable clamps, cotter pins	Condenser tubes	Bolts, nuts, gears, pinions, valve bushings, high strength forgings, valve bodies and stems, tie bolts, marine hardware, sucker bolts, hardware bushings
General properties	Corrosion resistance of copper, mechanical properties of mild steel	Tensile properties comparable to cartridge brass, corrosion resistance similar to that of copper, welding properties only slightly inferior to 420, nonmagnetic	High physical properties, high resistance to acids, most resistant of bronzes to H_2S	Unusually high tensile strength, excellent corrosion resistance, readily hot forged, rolled and extruded, free machining
Working properties:				
Cold working	Good	Excellent	Good	Poor
Hot working	Excellent	Do.	Do.	Excellent
Machining	Fair	Fair	Fair	Do.
Welding	Gas, carbon arc, metal arc, spot and seam for thin gage	Gas, carbon arc, metal arc, spot and seam for thin gage	Gas, carbon arc, metal arc	Gas, carbon arc, metal arc, spot and seam for thin gage
Soldering	Excellent	Excellent	Good	Fair
Polishing	Good	Fair	Do.	Excellent
ASTM specifications:				
Sheet	B96, B97, B100	B97		
Rod	B98, B99, B124	B98, B99, B105		B124, B150
Tube			B111	

BRONZE ALLOYS BY ALLOY TYPE, ALLOY NAME, COMPOSITION, USES, PROPERTIES, SPECIFICATIONS /a

Alloy type	Tin Bronze		
Alloy name	Phosphor Bronze, 5% - Grade A	Phosphor Bronze, 8% - Grade C	Bearing Bronze
Chemical composition:			
Copper	Balance	Balance	90.0
Zinc			9.5
Phosphorous	0.03-0.35	0.03-0.35	
Tin	3.5-5.8	7.0-9.0	.5
Typical uses	Diaphragms, bellows, lock washers, cotter pins, fuse clips, clutch disks, springs, screw machine stock, expansion plates	Springs, perforated sheets, bellows, cotter pins, fuse clips, bushings, lock washers	Bushing material for light loads, weatherstrip applications, fuse clips, lamp connections
General properties	High tensile strength, high resistance to corrosion and fatigue, low coefficient, high immunity to season cracking	High corrosion and fatigue resistance, low friction coefficient, high tensile strength	High physical properties, resistant to atmospheric corrosion and tarnish, low friction coefficient
Working properties:			
Cold working	Excellent	Excellent	Excellent
Hot working	Poor	Poor	Excellent if lead-free
Machining	Fair	Fair	Fair
Welding	Gas, carbon arc, metal arc, spot and seam for thin gage	Gas, carbon arc, metal arc, spot and seam for thin gage	Gas, carbon arc, metal arc
Soldering	Excellent	Excellent	Excellent
Polishing	Do.	Do.	Do.
ASTM specifications:			
Sheet	B100, B103	B100, B103	Tests on 325 conducted on 0.040-in. stock. Basis of rating: hard on material previously rolled 6 B & S nos. Grain at ready to finish, 0.080-0.150 mm. soft-annealed at 1,300°F. for 1 hour
Rod	B139, B159	B139, B159	
Tube			

a/ Source: American Society for Metals, Metals Handbook: 1948 ed.

Copper-Nickel Alloys

Copper and nickel are soluble in each other in all proportions. They form a continuous series of useful alloys, especially on the copper side ranging from 95-percent copper and 5-percent nickel used for the driving bands on projectiles to the 60-percent copper and 40-percent nickel alloy called Constantan, of high electrical resistance, used as resistance wires in such instruments as potentiometers. Besides these simple binary alloys there are many compositions embodying such additional elements as iron, tin, zinc, aluminum, and silicon that serve a great variety of special purposes both with respect to electrical properties and the general field of strength properties including fatigue resistance, hardness, etc.

The following table lists the principal copper-nickel alloys.

NICKEL ALLOYS BY NAME, COMPOSITIONS, USES, PROPERTIES, SPECIFICATIONS /a

Alloy name	Nickel Silver, 18% Deep Drawing	Nickel Silver, 18% Spring Stock	Nickel Silver, 15%	Nickel Silver, 10%	Nickel Silver, 12% Leaded	Cupro Nickel, 10%	Cupro Nickel, 30%
Composition:							
Copper	65.0	55.0	66.0	66.0	65.0	88.5	70.0
Zinc	17.0	27.0	19.0	24.0	20.7		
Lead					2.0		
Nickel	18.0	18.0	15.0	10.0	12.0	10.0	30.0
Manganese					0.3	Fe 1.5	
Typical uses	Marine and automotive trim, hardware, architectural panels, lighting, electrical and plumbing fixtures camera parts, various equipment in the process industries, slide fasteners, tableware, jewelry, stamping, etchings	Springs, marine and automotive trim, hardware, architectural panels, lighting, electrical and plumbing fixtures, various equipment in the process industries, slide fasteners, tableware, jewelry, stamping, etchings	Hollow ware, panel sheet to match complimentary trim	Inexpensive jewelry, hollow-ware (dishes and trays), stampings, embossing decorative trim	Keys, products requiring machinability, lock washers, cotter pins, fuse clips	Condenser tubes, pipe and tube	Condenser tubes and plates, tanks, vats, vessels, process equipment, automotive parts, nuts, bolts, screws, meters, refrigerator pump valve
General Properties	High physical properties, high resistance to corrosion and tarnish, malleable and ductile, Color: Silver-blue-white	High physical properties, high resistance to corrosion and tarnish, high fatigue strength, Color: Blue-white	High physical properties, high resistance to corrosion and tarnish, malleable and ductile, Color: White	Ductility, deep drawing Color: Yellow-white	Good machinability, prefer nonleaded for bending and drawing Color: White	Very resistant to corrosion and erosion, particularly in salt or brackish water	High strength and physical properties, high ductility, resistant to corrosion and erosion, Color: White-silver
Working Properties:							
Cold working	Excellent	Good	Good	Excellent	Poor	Excellent	Excellent
Hot working	Fair	Fair	Fair	Fair	Fair	Good	Fair
Machining	Do.	Do.	Do.	Do.	Excellent	Fair	Do.
Welding	/b	/b	/b	/b	Nonleaded prfd.	Gas, metal arc, resistance	Gas, metal arc, resistance
Soldering	Excellent	Excellent	Excellent	Excellent	Excellent	Excellent	Excellent
Polishing	Do.	Do.	Do.	Do.	Do.	Do.	Good
ASTM specifications:							
Sheet	B122	B122		B122			
Tube							B122, B171 B111

a/ American Society for Metals, Metals Handbook: 1948 ed.
b/ This alloy can be soft-soldered, silver-soldered, bronzed, oxyacet, line-welded, carbon arc-welded, and resistance-welded.

Other Alloys

In addition to the alloys already mentioned, copper forms many useful alloys with other elements, most notable of which is beryllium.

Beryllium-copper alloys, containing 2 to 4 percent beryllium, are noted for their toughness, hardness, tensile strength, fatigue resistance, low sparking, and nonmagnetic properties. Beryllium is the most effective hardening agent known for copper; the addition of 2.25 percent beryllium produces an alloy which, after heat treatment, develops a strength, in pounds per square inch, six times that of commercial copper.

Beryllium-copper alloys are used for springs, nonsparking tools, plastic molds, and strong mechanical parts. The alloys are becoming increasingly important in aircraft-component assemblies, communications equipment, signaling devices, and various ordnance accessories.

The master alloy of beryllium-copper, containing about 4 percent beryllium, and from which other alloys are prepared, is made by arc-furnace fusion of a mixture of beryllium oxide, carbon, and copper powder or chips. Silicon is sometimes added to further harden and strengthen the alloy.

Copper is also used widely as a minor constituent in many other alloys. Nickel-copper alloys are an important group in this category. One of the most important nickel-copper alloys is "Monel metal," a "natural" alloy produced directly from Canadian Bessemer matte which has been made from nickel-copper ore of the Sudbury, Ontario region. This alloy was introduced in 1905 by the predecessor company of International Nickel Company of Canada, Ltd. The average composition is nickel, 67 percent; copper, 28 percent; and iron, manganese, silicon, and other elements, 5 percent. The alloy may be cast, rolled, or forged, and can be annealed after cold working. It is resistant to corrosion and to the action of many acids and will retain its bright nickel-white surface under ordinary conditions. Monel metal is employed for parts for chemical and mining equipment, marine fittings, kitchen and restaurant equipment, and valves. Mond metal, a synthetic alloy of this type, was developed by the American Nickel Corp. Its composition is 70 percent nickel, 26 percent copper, and 4 percent manganese. Other nickel-copper alloys, essentially of the Monel-metal type, designated as S Monel, K Monel, R Monel, H Monel, and Ebonized Monel, contain varying amounts of other elements - aluminum, iron, sulfur, and carbon - and are used principally for valves.

Copper in the proportion of about 4 percent, with fractional percentages of silicon and magnesium, imparts to aluminum the strength and hardness of mild steel.

Zinc-base alloys for die-casting usually contain copper in some proportions; and tin-base antifriction metals almost invariably contain 3 to 5 percent copper as an essential ingredient. Copper-steel alloys, generally containing less than 1 percent copper, are employed for construction work where mild resistance to corrosion is needed and where the cost of the higher-resistant chromium steels is not warranted.

The precious metals, gold and silver, usually are alloyed with copper. United States coins contain 10 percent copper, which imparts hardness and toughness to the coins without rendering them brittle. Jewelers' gold may be an alloy with copper or silver, or with both. "Sterling" silver consists of 92.5 percent silver and 7.5 percent copper.

3. Copper Compounds

In point of tonnage, copper sulfate ($CuSO_4$), containing about 40 percent copper, is the most important to industry of all the copper compounds. Anhydrous copper sulfate is a white crystaline substance, but the usual commercial form is the pentahydrate ($CuSO_4 \cdot 5H_2O$), called blue vitriol which contains about 26 percent copper. Domestic agricultural use accounted for about 60 percent of the 73,000 tons of copper sulfate produced in 1949. It is highly valued as a fungicide and as a source of minerals for plant and animal life. Copper sulfate is also used extensively as a depressing and dispersion agent in metallurgical flotation plants, as a print toner in photography, in dyes, galvanic cells and antiseptics, also as the raw material in the production of the complex copper ammonium compound necessary for making crayon. Other copper compounds and their uses are given in the following table.

COPPER COMPOUNDS AND THEIR USES

Compound name	Formula	Crystal color	Soluble in	Typical uses
Basic cupric acetate	$Cu(C_2H_3O_2) \cdot CuO \cdot 6H_2O$	Blue	Water, ammonium hydroxide, acid	Catalyst for making organic compounds.
Basic cupric carbonate	$CuCO_3 \cdot Cu(OH)_2$	Dark green	Slightly in water and acid	Raw material for production of other copper compounds, pigments for paints and ceramics, fungicide and pyrotechnics.
Cuprous chloride	$CuCl$	White	Hydrochloric acid, ammonium hydroxide	Catalyst in chemical manufacturing, condensing agent for soaps, fats, and oils.
Cupric chloride	$CuCl_2$	Yellow brown	Water, alcohol, ammonium chloride	Mordant, refining of gold and silver, recovery of mercury from ores, pyrotechnics and photography.
Cupric nitrate	$Cu(NO_3)_2$	Blue	Water, alcohol	Electroplating solutions, burnishing iron, preparation of metal catalysts.
Cuprous oxide	Cu_2O	Dark red	Hydrochloric acid	Production of copper salts, ceramics, electroplating, fungicides, antifouling paints.
Cupric oxide	CuO	Brownish-black	Acids	Rayon and ceramic industries, electrical depolarizer, chemical analysis, catalyst.

C. MINERALS AND ORES

Mineralogically, copper ores are divided into three groups, namely native copper ores, oxide ores, and sulfide ores. In the United States the sulfide ores yield about 85 to 90 percent of the total primary production.

1. Minerals

Native copper occurs in most of the principal copper deposits of the world, but usually in small quantities. It has been found in 27 States of the United States, in Bolivia, Chile, Australia, and elsewhere. The deposits of the Lake Superior district and Corocoro, Bolivia, are the only ones of economic importance in which metallic copper is the chief ore mineral. The native metal is very pure, containing 98 to 99.2 percent copper, with small amounts of silver mechanically enclosed and not alloyed, and minor quantities of arsenic.

Cuprite, malachite, azurite, and chrysocolla are the chief minerals of the oxidized zones of copper deposits. Brochantite and atacamite are found only in arid regions and in the upper portions of the orebodies.

Chalcocite and chalcopyrite from which the great bulk of copper production is derived are sulfides of wide geographic distribution.

The principal copper minerals are as follows:

Mineral	Composition	Specific gravity	Copper, percent
Native:			
Native copper	Cu	8.8-8.9	100
Oxides:			
Cuprite	Cu_2O	6.0	88.8
Tenorite	CuO	5.8-6.3	79.8
Malachite	$CuCO_3Cu(OH)_2$	3.9-4.03	57.3
Azurite	$2CuCO_3Cu(OH)_2$	3.77	55.1
Chrysocolla	$CuSiO_3 \cdot 2H_2O$	2.0-2.4	36.0
Antlerite	$Cu_3SO_4(OH)_4$	3.39	54.0
Brochantite	$Cu_4SO_4(OH)_6$	3.9	56.2
Atacamite	$CuCl_2 \cdot 3Cu(OH)_2$	3.75-3.77	59.4

Mineral	Composition	Specific gravity	Copper, percent
Sulfides:			
Chalcopyrite	$CuFeS_2$	4.2-4.3	34.5
Bornite	Cu_5FeS_4	4.9-5.4	63.3
Chalcocite	Cu_2S	5.5-5.8	79.8
Covellite	CuS	4.9-5.0	66.4
Enargite	$Cu_3As_5S_4$	4.43-4.45	48.3
Tetrahedrite	$Cu_8Sb_2S_7$	4.7-5.0	52.1
Tennantite	$Cu_8As_2S_7$	4.7-5.0	57.0

2. Ores

Ore is defined as an aggregate of minerals from which one or more mineral products may be extracted profitably. This definition thus includes not only mineral in its natural place in the earth's crust but also mine dumps, tailing piles, and so forth, which can be reworked at a profit. No distinction is made between metals and nonmetals. The consideration of commercial extraction implies that the unworkable materials of today may become the ore of tomorrow if decreased costs or increased prices enable marginal ores to be treated profitably.

Copper ores are found in nearly all types of ore deposits in igneous, sedimentary, and metamorphic rocks and may be of either primary or secondary origin. The primary minerals were deposited during the original period or periods of metallization; secondary minerals are alteration products of primary minerals as a result of weathering or other surficial processes resulting from descending surface waters.

In comparatively arid regions, where copper-bearing sulfides are disseminated in intrusive igneous rock, descending ground waters develop an upper leached zinc, underlain by a secondary chalcocite-bearing section of high copper content. Thus have originated some of the porphyry copper ores now being mined extensively in the southwestern United States.

Copper ore-deposits may be classified by many different methods. A common classification, based upon mineralogy and metallurgical treatment, with examples of deposits, is included here.

1. Sulfide ores:
   a. High-grade, direct-smelting ores: Katanga, Belgian Congo; Northern Rhodesia.
   b. Medium-grade ores which must be concentrated: Butte, Mont.; Cerro de Pasco, Peru; O'Okiep, Southwest Africa.
   c. Low-grade ores, which require concentration and must

be mined and milled on a large-scale, low-cost basis: Morenci, Ariz.; West Mountain (Bingham), Utah; Ajo, Ariz.; Cananea, Mexico; Braden and Chuquicamata, Chile; Mansfeld, Germany.

d. Pyritic ores: Rio Tinto, Spain; Outokumpo, Finland.

2. Oxidized ores:
   a. High-grade or medium-grade ores which can be smelted to "black copper" by reduction smelting, mixed with sulfide ore or concentrate for matte smelting, or leached: Katanga, Belgian Congo; Northern Rhodesia.
   b. Low-grade ores which are treated by leaching: Inspiration, Ariz.; Chuquicamata, Chile; Ajo, Ariz.

3. Native copper ore: Lake Superior, Mich.

Under 1c and 2b appears the group of copper ore-deposits which is the most important of all economically, the "porphyry coppers". These are "disseminated copper deposits," in which the copper minerals in the form of small grains are scattered uniformly through the large body of rock. The copper minerals in the upper portions are in general oxidized, and those lower down are sulfides. Parsons (3) lists the following characteristics of the porphyry copper deposits.

1. The deposit is of such magnitude and shape that it can be mined advantageously by large-scale methods, either by underground caving or in open pits.

2. The copper minerals are distributed so generally and uniformly that "bulk" methods of mining are more profitable than selective methods whereby individual veins or thin beds would be stoped separately.

3. An intrusion of porphyry or closely related igneous rock has played a vital part in the genesis of the ore, though the porphyry itself may not constitute the major part of the deposit.

4. The process known as "secondary enrichment" has usually operated to concentrate the copper.

5. The extent of the ore body is usually determined by economic limits rather than by geologic structure, because the copper content gradually diminishes as progress is made either downward

or laterally from the core of an enriched mass. At some point - which necessarily varies with the cost of production at the particular mine, with the price of copper, and with other economic conditions - a "cut-off" must be made between "ore" and "waste". This may be 0.5 percent copper or it may be 1.5 percent in different mines; and, considered literally, it would vary widely with respect to the same mine at different times.

6. The average copper content of the mass is comparatively low (with 3 percent as the maximum), and grinding and mechanical concentration are necessary to produce a suitable smelter feed, if the ore is sulfide in character.

The ores of copper are mixtures of ore minerals and various waste minerals known as gangue, among which are rock matrix, quartz, calcite, dolomite, siderite, rhodochrosite, barite, and zeolite. Some copper ores are relatively free of contaminants, but the bulk of all copper ores also contain varying amounts of other elements, principally arsenic, antimony, bismuth, iron, lead, zinc, gold, silver, molybdenum, nickel, cadmium, and cobalt. Many zinc, lead, and silver ores contain copper, which may be recovered as a by-product, and in complex ores copper may also be associated with zinc and lead.

The type of copper ore and its tenor, that is percentage of copper content, generally determine the method of treatment. The lowest-grade ores are the large deposits of disseminated sulfides, known as "porphyries", and the native-copper deposits, which may contain as little as 0.8 percent copper. Sulfide ores now being mined by underground methods contain as little as 0.75 percent copper or 15 pounds per ton, of which 12 to 14 pounds per ton is recoverable. High-grade sulfide ores may range upward from 3 or 4 percent. Oxidized ores containing about 1 percent copper are now being mined in the United States. Ores carrying 5 percent or more copper are generally smelted directly to avoid concentration losses, provided facilities are available and the distance to the smelter is not too great. Ores with less than 3 percent copper are generally concentrated and the concentrates smelted. For the 5-year period 1945-49, the average recoverable copper content of all smelting ores in the United States was 3.52 percent, of all concentrating ores 0.89 percent, and of all ores 0.91 percent, compared with 4.03, 1.01, and 1.09 percent, respectively, for the preceding 5-year period, 1940-44.

The general trend towards mining and milling of low-grade copper ores since 1910 is illustrated in the following table.

Domestic Copper Mines

Year	Average tons ore per year, in millions of tons	Yield a/ copper percent
1910-19	42.4	1.65
1920-29	46.9	1.51
1930-39	32.6	1.46
1940-44	85.8	1.09
1945-49	77.7	.91
Northern Rhodesian b/ Copper Producers		
1939	7.4	3.72 c/
1947	7.7	2.87

a/ 1910-36 data from Leong, Y. S., and others, Copper Mining: National Research Project, Phila., 1940, p. 220. For subsequent years, from Bureau of Mines Minerals Yearbook.
b/ Roan Antelope, Rhokana, and Mufulira; from annual reports of respective companies.
c/ Grade of ore produced.

Profitable exploitation of low-grade copper ores is achieved by large scale, open-cut methods, using power machinery such as power shoevels, bulldozers, trucks, and locomotives, or underground by low-cost block-caving or other methods yielding high labor-efficiencies.

Arizona and Utah together supplied 74 percent of the copper produced in the United States in 1949, from ores averaging 0.91 percent and 0.89 percent recoverable copper, respectively.

A further discussion of grade and type of ore in relation to industry structure, labor productivity, technologic progress and related factors is given in chapter VI.

The mineralogy and general geology of ores in the principal districts of the world are given in chapter III, with ore-reserve estimates.

BIBLIOGRAPHY

1. American Society for Testing Materials Standards, Part 2, Non-ferrous Metals, 1949, pp. 315-349.

2. Malozemoff, Plato, The Real Danger of High Costs: Eng. and Min. Jour., vol. 150, November 1949, pp. 72-79.

3. Parsons, A. B., The Porphyry Coppers: Am. Inst. Min. and Met. Eng. (Rocky Mountain Fund), 1933.

II. MAJOR MINING METHODS AND METALLURGICAL PROCESSES

23535 O—52——6

A. PROSPECTING, EXPLORATION, AND DEVELOPMENT

Mining has been defined as "the art or practice of operating mines profitably". Mineral extraction differs in certain respects from most other industrial enterprises. The salient, distinguishing feature is the fact that mines are generally considered to be wasting assets, that is material that cannot be replaced is removed from the ground. It should be emphasized, however, that, where possible, most mines follow a policy of developing ore in quantities equal to the tonnages mined. This practice has been highly successful; present (as of January 1, 1950) ore reserves in the United States and in many other countries with significant copper deposits are comparable in tonnage with estimated reserves of 20 years ago, even though world mine production of metallic copper has totaled over 35 million tons during that time. This seeming anomaly has as its basis the discovery of new deposits and, more important, additions to the ore supply at existing workings by the ever-changing concept of what constitutes reserves, as brought about by price changes and technological advances. Another important factor in the conception of ore reserves is the fact that reserves are seldom proved in advance of 10 years; exploration work is relied upon to add new reserves as the ore is mined out. This practice does not, of course, always add tonnages equal to the quantities of ore removed; ore bodies ultimately become depleted.

Prospecting is the search for ore, and exploration is the work involved in gaining a knowledge of the size, shape, position, and value of an ore body. Development is the preparation of a mine for ore extraction.

1. Prospecting

Most of the world's metal mines were originally discovered by individual prospectors using such simple methods as tracing float, that is, tracing weathered minerals to their sources by panning, and digging shallow trenches or test pits. Hydraulicking or bulldozing and other scraping methods are used to strip shallow layers of overburden to expose bedrock for further examination.

The art of prospecting has developed as a series of separate and distinct techniques, each of which, as accepted, peaks rapidly, carries on for a few years, and then becomes relatively unimportant.

The techniques of individual prospectors, though flashes of genius in their conception, are so simple of application as to be understandable and usable to all who seek minerals intelligently. Panning for gold is the basic technique of the mineral prospector. Seldom has a living been made by gold panning alone, but almost all great lode and placer mines have been discovered and their

general outlines delimited by systematic panning to localize the areas of concentration. As the art of gold prospecting developed and the demand for other metals increased, prospectors learned to recognize familiar geologic and physical aspects of ore occurrence, to extend the gold-panning technique to the tracing of other heavy metals or their minerals to their source, and to sample and assay the exposed outcrops of suspected mineral deposits.

Surface prospecting, even with the aid of surface geology, is almost an exhausted technique for discovering deposits of the metals that have long been in active demand. The fluorescent lamp has passed its peak of usefulness, at least in the United States, in the search for some of the minerals of tungsten. The Geiger counter has been useful in finding radioactive minerals and may lead to additional discoveries before all promising areas of the earth's surface have been covered.

In the search for the common metals, including copper, lead, and zinc, individual techniques are now so unproductive that few prospectors are willing to take the risks or endure the hardships of extensive field work. Newer techniques are needed that are cheaper, better, or faster than existing methods. Future ore discoveries will be chiefly of deposits that do not outcrop, and these do not lend themselves to discovery by old-time methods. The efforts of the prospector today must be supplemented by scientific studies and systematic search by organized field parties using the latest geological and geophysical techniques.

In the past, mineral deposits have not been discovered by the major companies but by individuals, local groups organized to prospect and explore a single property, and small companies. The most-advanced techniques, with their expensive requirements of mapping, drilling, or underground workings, sampling and assaying, metallurgical testing, and other related activities, are beyond the financial capacity of individuals or small companies. The major companies, for the most part, find operation of small mines uneconomic and generally do not become interested in a mining property or a mineralized area unless it promises to become a large-scale producer.

Geophysical prospecting attempts to measure and interpret anomalous physical or physicochemical phenomena within the earth's crust. For example, magnetic material in the earth increases the strength of the earth's normal magnetic field, and the resulting magnetic anomaly can be measured with sensitive magnetometers, hence this method has been important in locating magnetic iron ores (magnetite) and has been helpful in tracing geological formations and certain types of nonferrous mineral deposits in which there is a greater or smaller content of magnetic minerals than in the enclosing or adjacent rocks.

The petroleum industry has been eminently successful in developing and applying gravitational and seismic methods to the location of underlying oil-bearing structures. The gravitational methods are based on gravity differences between different types of rocks, and the seismic methods depend upon differences in the speed of refracted and transmitted sound waves through different rock layers. Magnetic anomalies at borders of oil-bearing formations are also becoming useful indicators in petroleum prospecting.

Various other geophysical methods, such as resistivity and radio frequency, depend on electrical phenomena, but none of the methods thus far developed have been notably successful in locating hidden deposits of nonferrous metals.

A realistic approach to the problem of maintaining our mineral resources implies the expenditure of ever-increasing amounts for developing new techniques in coordination with detailed mapping and geologic studies.

In some connections, boring and underground methods are applied to prospecting, but since these are more commonly for the purpose of exploration they are discussed in the following section.

2. Exploration

The data obtained by exploration are required for determining ore reserves, planning of mining methods and equipment, projecting the scale of operations, and other technological and economic factors essential to establishment of mining enterprise.

These objectives postulate that the ore body be penetrated by boreholes and/or underground workings at appropriately spaced intervals and that the exploratory work be carried far enough to permit sound planning of subsequent operations. For example, the porphyry-copper deposits are generally explored by a network of boreholes or underground workings on a 100- to 200-foot spacing over the entire area of the deposit in advance of mining operations on a large scale, to predetermine the factors that predicate a heavy expenditure for equipment and to avoid placing surface structures on ground that may later be mined or caved. On the other hand, exploration of a vein deposit, especially if it is to be mined by supported stopes, need only be carried far enough to demonstrate the existence of a workable ore body of sufficient extent and value to justify relatively moderate expenditures for development and equipment. Deferred interest charges on advance exploration work and the cost of maintaining underground workings limit the extent of exploration that may be done in advance of mining, hence, in many underground mines the proved ore reserves do not exceed 10 years' life and often are less than 3 years. A continuing program of exploration is relied upon to add new reserves to replace ore mined out.

Boring

Boring is employed extensively, either as the principal exploratory method or to supplement exploration by underground and surface workings. Three principal types of drills are employed for this purpose - churn drills, core drills, and hammer drills. Each type of drill has its limitations and its particular field of usefulness.

Churn drills: In churn drilling a string of tools with a cutting bit at the lower end is suspended from a rope or cable and is alternately raised and dropped by hand or a power-driven mechanism, chopping a hole in the rock. Water is used in the hole during drilling, and the cuttings are removed with a bailer at about 5- or 10-foot intervals. These cuttings are used as a sample to be examined and assayed.

Churn drills are employed when solid cores are not required, stratigraphic thicknesses do not have to be measured accurately, and only vertical holes are desired.

Churn drills were used in exploration at Bisbee, Ariz., in 1913-14, where 60,000 feet of hole was drilled at $1.97 per foot at a total cost of $118,200. At Miami, Ariz., in 1910-14, 44,500 feet was drilled at $3.26 per foot, totaling $145,070. Present costs for holes up to 1,000 feet in depth range from $2.00 to $5.00 per foot.

Core drills: Two types of core drills are used in exploring metal-bearing deposits, diamond drills and shot drills. Both forms are designed to yield a core of the substance penetrated. Both are power-driven rotary drills that cut an annular groove about a central core of rock. Diamond drills use small diamonds (bortz) mounted in the cutting bit, and shot drills use loose chilled shot fed into the hole as required.

Diamond-drill bits were formerly set with a small number, generally 8 to 16 carbonados, or black diamonds ranging to over 2 carats 1/ in weight, but in modern practice most bits are now set with about 40 to 200 bortz, white diamonds too imperfect or too small for use as gem stones and generally less than 0.2 carat in weight. The high cost of carbonados and the highly skilled labor required to set them in bits, together with generally faster drilling speeds with bortz, have caused the change in practice.

1/ The carat weighs 205 mg., or 3 1/16 troy grains. The international metric carat (c.m.) weighs 200 mg.

Diamond drilling is widely used in exploration because of its speed and adaptability for directional drilling. The United Verde mine, Arizona, drilled 6,922 feet in 1937 at a cost of $1.19 per foot. Present costs for diamond drilling range from $2.00 to $8.00 per foot.

Chilled-shot drills have only limited use in prospecting but are of considerable value in exploration work, occasionally being used for drilling holes large enough to permit engineers or geologists to enter. Their use is limited to vertical holes.

Hammer drills: Hammer drills have their best application in exploratory drilling underground, where relatively short holes are to be drilled and core recovery is not required. The maximum depth of drill holes for hammer drills is about 250 feet, but the most efficient range is under 150 feet. These are common heavy rock drills, using sectionalized hollow drill steel and standard bits. Holes may be drilled at all angles from horizontal to vertical; but, on account of the weight of the steel, long holes are usually fairly flat.

Underground Exploration

Most prospects and small mines rely principally upon underground methods of exploration. Typical examples are found in the exploration of narrow veins by sinking a shaft in or below the vein, following its dip, or inclination from the horizontal, and then driving underground horizontal workings in the vein known as drifts, or levels, at convenient intervals. Exploration for parallel ore bodies or to find faulted segments of the main ore body is done by crosscuts, or horizontal openings cutting the enclosing country rock at an angle to the prevailing horizontal direction or strike of the vein or stratified country rock. In suitable topographic situations, a horizontal opening or adit may be driven into a hillside to reach or follow a vein that crops out at a higher level. If this opening is later continued on through the hill it becomes a tunnel. A connection from a lower to an upper level or to the surface is known as a raise, if made by upward advance. An inclined or vertical opening advanced downward from an underground working is a winze.

3. Mine Development

The term "mine development" is employed to designate the operations involved in preparing a mine for ore extraction. These operations include tunneling, sinking, crosscutting, drifting, and raising.

In most mines, both exploration and development continue after ore extraction has begun and often nearly to the cessation of mining.

Although exploration and development work are similar, the emphasis is placed on ore finding in the former, whereas in the latter operations are directed mainly toward preparation for economical removal of the ore from the mine.

The major operations in developing an open-pit mine are stripping, which is the removal of barren or low-grade overburden, and the establishment of surface transportation.

The development of an underground mine involves the following: An efficient entry, whether a shaft or adit; an auxiliary entry, required for safety considerations if workings extent beyond depths as specified by various State laws; and such networks of drifts, cross-cuts, and raises (or winzes) at various levels and intervals as may be required by the form of the deposit and the details of the mining method to be employed. Transportation systems for removal of ore to the treatment plant and pumping of infiltrated waters are common to both surface and underground operations.

The cycle of operations in metal-mine development consists of drilling, blasting, and removal of broken ore. Drilling is generally done with compressed-air hammer drills, or, more rarely, with diamond drills. Blasting is done with various types of dynamite, detonated by electric caps or by fuses and caps.

Compressed-air or electric shoveling and loading machines are now used for removal of broken rock and loading in underground headings and shafts in most important underground operations, replacing hand shoveling. Power scrapers are used in gently inclined raises, and gravity in steeply inclined (usually over 38°) raises for transfer of broken rock to mine haulageways. Underground horizontal transportation for removing ore and waste rock is commonly done by car and track haulage, using storage-battery locomotives for short distances beyond the range of economic hand tramming and either storage-battery or trolley locomotives for longer distances. Ore is also transported by shuttle cars, Diesel trucks, and conveyor belts.

Some companies distinguish an additional step of preparation of underground mines between development and mining to include all of the auxiliary workings directly related to the preparation of a given block of ore for mining. "Stope preparation" is then confined to main entries and haulageways. This is an important distinction in the mining of massive deposits covering an extensive horizontal area, as in the porphyry-copper mines to be mined by caving methods, where advance preparation of stopes is a definite and extensive phase of operation that can be charged against a specific block of

of ore; but in narrow veins, thin beds, and relatively small irregular ore bodies there is generally no essential difference between development and preparation.

B. MINING

From the viewpoint of mining methods, there are two principal classes of copper ore deposits, those in veins and those in large bodies of ore widely disseminated in the country rock, generally known as porphyry ore. In general, veins and other deposits of tabular or lenticular form are mined by underground workings, employing methods aimed at complete extraction of workable ore and support of the workings to prevent loss of adjacent or overlying ore bodies and destruction of surface structures. The porphyries are mined by open-cut power-shovel methods if they lie close enough to the surface that the cost of removing the overburden is not excessive. If the overburden ratio is too high, these deposits are generally mined by caving systems of mining which are amenable to mass-production operation at minimum cost.

In recent years improvements in underground methods have consisted mainly of modifying existing techniques. Progress has been reflected in the growth of mechanization. Improved drilling equipment, loading equipment, and mechanical ventilating systems have been introduced.

The principal mining methods employed in modern copper mining are: (10)2/

1. Open pit.
2. Caved stopes.
3. Supported stopes.
    a. Naturally supported.
    b. Artificially supported.
        i. Shrinkage stopes.
        ii. Cut-and-fill stopes.
        iii. Timbered stopes.

The following table shows the relative importance of the major groups of copper-mining methods and brings out the progressive increase in the application of open-pit methods at the expense of underground methods.

2/ Numerals in parenthesis refer to items in the selected references section of the bibliography at the end of the chapter.

PRODUCTION OF COPPER ORE AND RECOVERABLE COPPER FROM OPEN-PITS AND UNDERGROUND OPERATIONS,
by percentages, 1939-50

Ore

	1939	1940	1941	1942	1943	1944	1945	1946	1947	1948	1949	1950
Open-pits	59	61	63	66	69	68	68	66	73	76	78	81
Underground methods	41	39	37	34	31	32	32	34	27	24	22	19

Recoverable Copper

Open-pits	41	44	47	51	54	57	61	58	68	68	70	74
Underground methods	59	56	53	49	46	43	39	42	32	32	30	26

1. Open-Pit Mining (7)(11)

Open-pit methods are applicable to mining ore deposits that apex at or near the surface. If the apex is below the surface, the overburden and barren capping overlying the ore must be removed in advance of open-pit mining. Removal of this material is known as stripping and is part of development work.

The stripping-pit limits must be extended beyond the limits of the ore pit to provide a bench, and the pit-wall slopes must be such as to prevent sloughing of overburden into the ore area. Where the ore is to be hauled from the pit by locomotive or trucks, additional excavation may be required to provide an approach of a grade suitable for hauling purposes. (Fig. 1.)

The choice between open-pit and underground mining of a given ore deposit is based theoretically on the ultimate estimated profit to be made, which, in turn, is based upon a number of factors, such as size, shape, and depth of ore body; relative costs of mining by the open-cut and by an underground method applicable to the deposit; dilution of ore with waste and ultimate recovery of ore; topography and surface improvements; climate and snowfall; availability of skilled labor (for underground mining); probable continuity of operation; and available capital. The stripping-ore ratio is a basic factor used for determining whether to employ open-pit or underground methods and for determining economic limits of open-pit mining. Before the late 1930's the average economic ratio of waste to ore was approximately 1:1, but improved equipment and practices in open-pit mining have reached the point where open pits may now (1949) be operated more economically than underground methods with waste to ore ratios as high as 3:1.

Among the advantages of open-pit mining are its flexibility, the ability to obtain mass production with it, and the ease with which rate of production can be increased or decreased once the pit has been developed fully; small shut-down expense; the ability to mine selectively to meet requirements for certain grades of ore (except in glory-hole or milling pits); complete extraction of the ore inside the pit limits (except as limited by ore benches for haulage tracks and maintenance of safe pit slopes, but this ore can usually be recovered finally by employing special clean-up methods); comparatively small number of men employed, a large proportion of whom are skilled labor in mechanized operations; and elimination of hazards inherent in underground mining operations.

FIGURE II-I
OPEN-PIT MINING

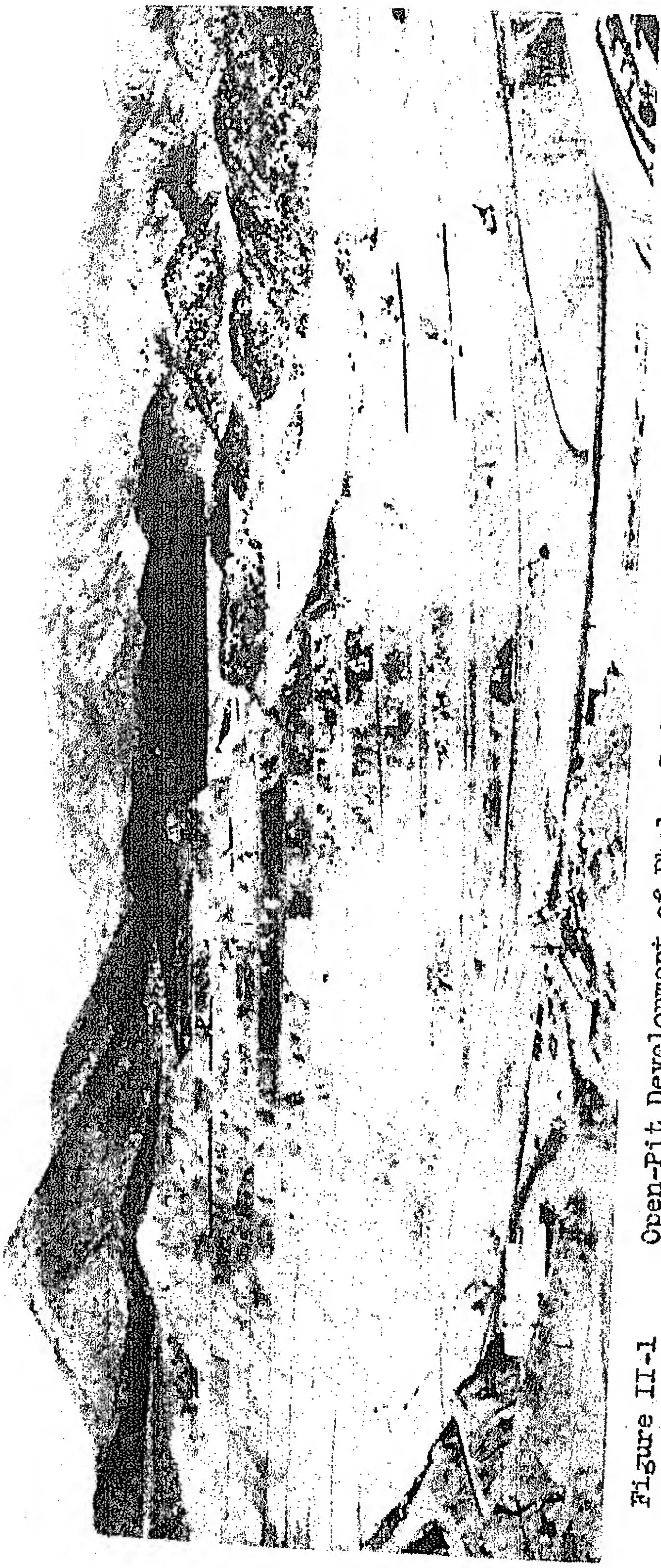


Figure II-1   Open-Pit Development of Phelps Dodge Corp. property
on Clay Mountain, Morenci, Arizona
(From Mining and Metallurgy, vol. 23, May, 1942, p. 241)

On the other hand, certain disadvantages may outweigh the advantages and affect direct economic considerations in some instances. Thus, large open-pit operations involve heavy capital outlay for equipment, and where the amount of overburden to be removed is extensive, correspondingly high capital expenditure is required for stripping. This capital is nonproductive until ore mining is begun, and during the stripping period heavy interest charges often accumulate. The time elapsing before production begins may in itself be a serious disadvantage, especially if exploitation is undertaken when ore prices are favorable and the demand for ore is strong. Disposal of the waste from stripping operations sometimes occasions a serious problem, especially when the terrain is flat or exorbitant prices must be paid for dump area near the mine. Climatic conditions may limit the operation to being seasonal and necessitate cessation of operations during certain months, and in areas where torrential rains are prevalent frequent flooding of the pit may be a serious obstacle to pit mining.

In 1950, approximately 75 percent of the total copper and 80 percent of the copper ore produced in the United States (see table on page 10) came from open pits, and in many other important copper regions elsewhere in the world (Chile, the U.S.S.R., Mexico) open-cut methods are widely employed.

Since late in the last century, power shovels and draglines for digging and loading and locomotives and cars for hauling from the pit have been in general use at most large-scale open-pit operations. The size and capacity of equipment have been increased through the years. Design and efficiency of the equipment and methods and equipment for drilling and blasting have been improved, and advances have been made in other practices as operations have been undertaken on a larger and larger scale and as the depths of stripping and ore pits have become greater.

Steam shovels and steam locomotives were early replaced by Diesel or electric motive power, and old railroad-type shovels have been replaced by revolving caterpillar-traction shovels driven either by electrical or Diesel power.

Track shifting and grading, spreading of waste on the waste dumps and clean-up of stripping on top of the ore were formerly performed manually. This work is now done principally by locomotive cranes, bulldozers, and dirt spreaders.

Bank drilling, formerly done by hand, long-handled shovels, or posthole diggers in soft ore or by piston drills is now done principally with mobile, caterpillar-mounted, electric churn drills from the top of the bank along the berms or benches of the pit.

More recently, motorized pit haulage has been adopted for the smaller pits and for clean-up work in larger pits where adverse grades are too severe for locomotive haulage. The first use of trucks for large-scale open-pit haulage in metal mines in the United States was at the United Verde mine.

Although a number of open-pit iron mines have installed conveyor belt systems for transporting the ore from gathering points in the pit to bins on the surface outside the pit limits, so far as is known none of the open-pit copper mines have applied this method of transportation.

As open-pit operations are extended to depths at which the stripping ratio becomes uneconomic, the glory-hole mining method has been employed in some cases. This method has also been used extensively for mining relatively small ore bodies or the upper parts of bodies that apex at or near the surface.

In glory-hole mining, the ore is broken down around one or more raises or millholes extending upward from an underground haulageway driven below the ore or beneath the ultimate bottom of the glory-hole or pit. It is thus a combination of surface and underground mining. The ore breaking is done on the surface but is drawn off through underground workings. Under favorable conditions, the glory-hole method is nearly as economical as open-pit mining for the same scale of operations.

As an index of the cost of open-pit mining compared to other methods, the average grade of ore produced by the open-pit copper mines ranges from approximately 0.80 to 1.00 percent. On the other hand, mines employing various underground methods in vein systems (excludes caving) require a grade of 3 to 6 percent to operate profitably. Underground caving methods operate efficiently on ores averaging as low as 0.80 percent copper.

2. Caved Stopes

Caved-stope systems of mining were originally applied on the iron ranges of Minnesota and Michigan and were extensively developed at a number of the porphyry-copper properties in the southwest, to mine low-grade ore bodies that lie at depths from the surface below economic stripping ratios of open-pit mining. At present, approximately 12 percent of the copper production of the United States is mined by caving systems.

When caved-stope methods are employed, breaks to and subsidence of the surface will occur ultimately if caving is continued over an area wide and thick enough in relation to the depth of cover. Hence,

caved-stope methods are applicable only where there is no objection to caving the overlying strata or to surface subsidence.

Caved stopes are of two distinct types: In the first, the ore is broken by caving induced by undercutting a block of the ore and isolating it from or weakening its connection to the surrounding ore or walls; in the second, the ore itself is broken, by conventional drilling and blasting methods, in a series of horizontal or inclined slices, and the capping is allowed to cave into and fill the space occupied previously by the ore.

Both types involve extensive preparatory work in advance of actual mining of ore. Working shafts of sufficient capacity are provided for hoisting ore to the surface and the transfer of men and materials. In block caving, main haulageways are driven on a level sufficiently below the bottom of the ore so that the branching transfer raises may be installed to connect with a sublevel on which the actual drawing off of ore is controlled by grizzlies or by car haulage or from a network of closely spaced draw raises that immediately underlie the caved ore. An example of the system of haulage drifts, transfer raises, and other workings for a block-caving system is shown in figure 2,(12) illustrating one of several variants of caving systems that have found favor in the underground porphyry mines of the southwestern United States.

Throughout mining an ore body of given horizontal area by a caving method, the withdrawal of ore must proceed in a systematically controlled manner to cause uniform settling of capping to avoid loss of ore in future operations by contamination of waste or by isolation of unmined blocks of ore. Hence the maximum rate of production from a given ore body depends largely upon a maximum horizontal area of the deposit and the extent of advance preparation. The rate of production cannot be increased substantially without a considerable time lag and in most cases some loss of efficiency due to overcrowded underground haulage and hoisting facilities that could be alleviated only by sinking and equipping a new working shaft with connecting underground haulageways.

Top slicing is an important modification of caving, whereby the ore is extracted by excavating a series of horizontal or inclined timbered slices alongside each other, beginning at the top of the ore body and working progressingly downward. Each slice, when mined out, is caved by blasting out the supporting timbers or allowing them to crush, bringing the capping or overburden down upon the bottom of the slice which has previously been covered with a floor or timber mat to separate the caved capping from the solid

FIGURE II-2
BLOCK CAVING

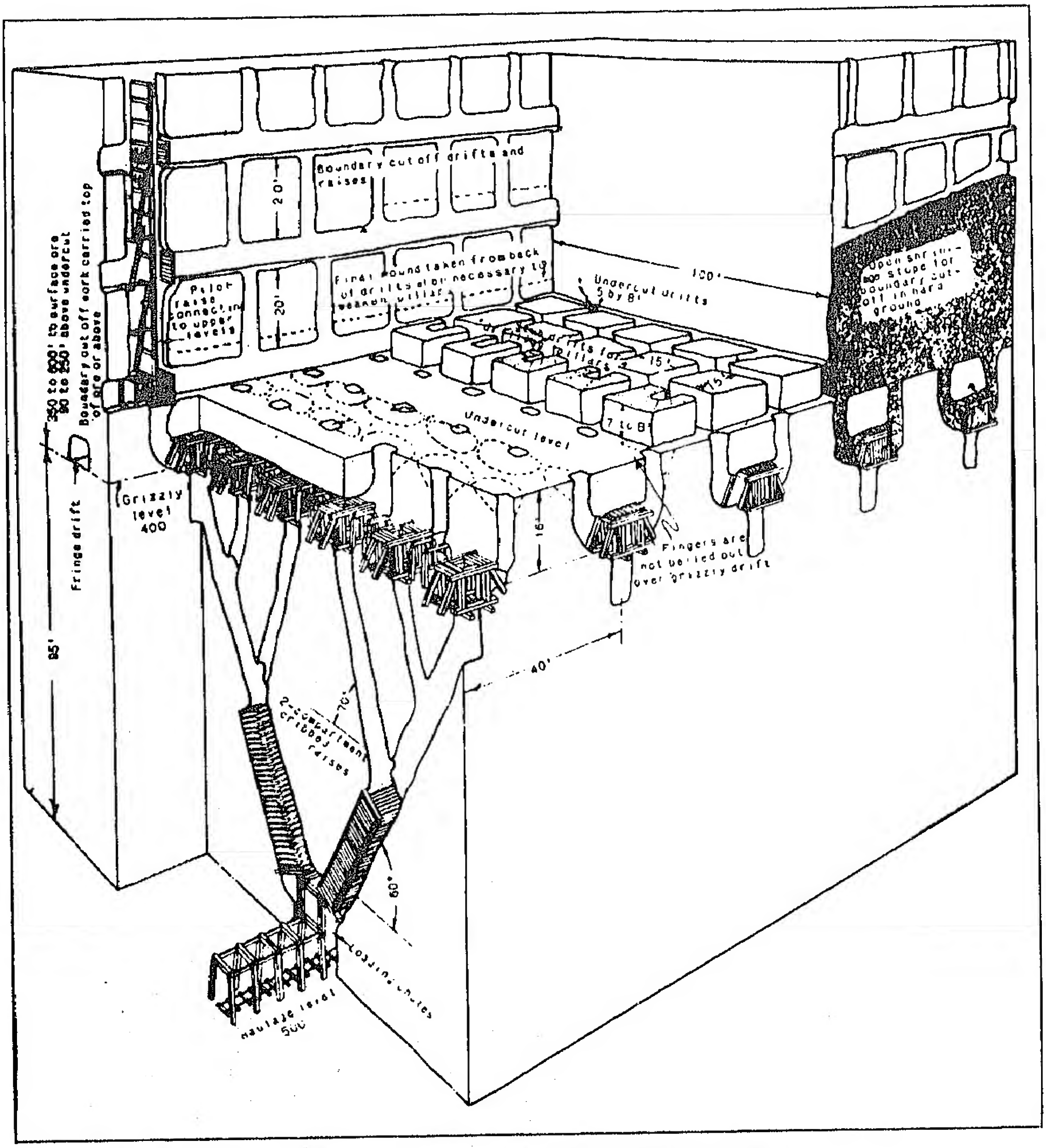


Figure II-2 Perspective of undercut block-caving method at Copper Queen mine
(From Bureau of Mines, Info. Circ. 6350, Oct. 1930)

ore beneath and to prevent admixture of waste with the ore. As successive slices are mined and caved, this mat follows the mining downward, filling the space formerly occupied by the ore. The mat also controls the movement of the caved overburden and prevents dilution of ore by barren capping.

The method has been applied most commonly in mining wide deposits of soft or weak ore overlain by a friable capping or unconsolidated overburden. A capping that breaks in large blocks that will wedge and arch over, leaving open holes in the mat, would be dangerous for top slicing. Hence, top weight or vertical pressure is an essential condition for successful top slicing.

The block-caving methods result in a certain loss of ore and a moderate dilution of ore with waste, but top slicing, although more costly, is capable of virtually complete ore recovery without dilution, hence is favored for ore bodies of somewhat higher copper content than those to which the block-caving methods are applicable. Top slicing is also more applicable to smaller and more irregular ore bodies than is block caving. It requires more highly skilled miners and requires more manual labor and more labor expenditures per unit of production.

3. Supported Stopes

The term "stoping" is employed in its broader sense to mean operation of excavation of ore by means of a series of horizontal, vertical, or inclined workings in veins or large irregular bodies of ore or by rooms in flat deposits. It covers the breaking of ore and its removal from underground workings, except those driven for exploration and development, and the timbering or filling of the stopes for the purpose of support.

Basically, the stoping method or methods that can be applied to a given ore body depend on the requirements for support of the stope, the maximum area or span of back and walls that will be self-supporting during the removal of the ore; the nature, size, and interval between supports required to maintain the backs and the walls of the overlying and surrounding country rocks and overburden to prevent their movement and subsidence. Variations of the principal methods of stoping may be based upon the direction or angle of workings, sequence of operations, or methods of handling the broken ore.

Naturally Supported Stopes

Stopes naturally supported are those in which no regular artificial method of support is employed, although occasional props, cribs, or stulls may be used to hold local patches of insecure ground.

The walls and roof are self-supporting. The simplest form is the open stope, in which the entire ore body is removed from wall to wall without leaving any pillars. It is applicable to relatively small ore bodies, as there is a limit to the length of unsupported span that will stand without support, even in the firmest and strongest rocks.

In open stopes with pillar support, the length of unsupported span is reduced by leaving pillars of ore, position and size being determined by localized ground conditions. It is frequently possible to leave low-grade ore within the ore body as pillars, making possible more complete recovery of the higher-grade ore.

Artificially Supported Stopes

Artificially supported stopes are those in which systematic temporary or permanent support of working faces and mined-out areas is provided by extraneous means.

Shrinkage stoping: In shrinkage stoping the ore is mined in successive flat or inclined slices, working upward from a level or the bottom of the block of ore. After each slice or cut, enough broken ore is drawn off from below to provide a working space between the top of the pile of broken ore and the back of the stope. (Fig. II-3.) Usually about 35 to 40 percent of the ore will be drawn off during active mining in the stope. The remaining ore serves as a floor upon which to work in drilling the back for succeeding cuts and also provides some support of a temporary nature to the stope walls. For this reason, shrinkage stopes are considered a form of artificially supported stope.

When active mining has been completed to the level above or to the floor pillar, the rest of the broken ore is drawn off from below, leaving the stope empty. It may be filled with waste later to prevent general movement and subsidence or to permit mining pillars left between stopes during the first mining.

Shrinkage stoping is applicable to bodies of strong, firm ore enclosed between firm walls that will not slab or slough off to any great extent after standing for a considerable time. The method is applied most frequently to relatively thin, tabular deposits dipping at angles greater than 50° in which few waste inclusions occur and which have fairly regular walls.

One of the serious disadvantages of shrinkage stoping is the delayed recovery of broken ore that is not drawn off until the entire block has been mined. Working conditions are not good, and

FIGURE II-3
SHRINKAGE STOPE

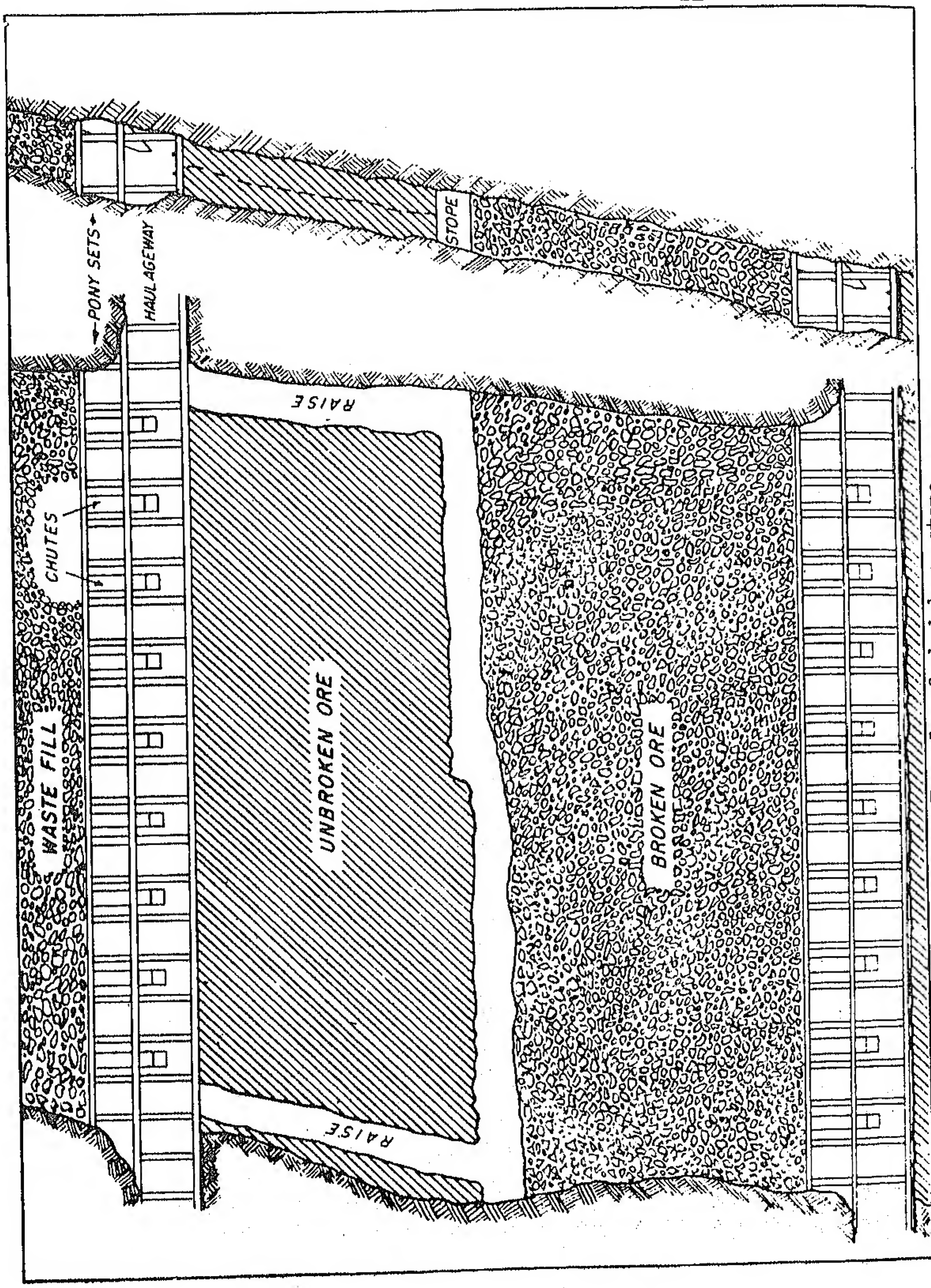


Figure II-3 Example of shrinkage stope
(From Bureau of Mines, Info. Circ. 7560, March 1950)

there is also more oxidation of sulfide ores than in systems that provide for immediate withdrawal of ore, which may adversely affect metallurgical recovery or, in extreme cases, result in a mine fire.

Cut-and-fill stopes: In cut-and-fill stoping the ore is excavated by successive flat or inclined cuts or slices, working upward from the level as in shrinkage stoping; but, after each cut, all the broken ore is removed and waste rock, sand, or other filling material is run in to within a few feet of the back, providing permanent support to the walls (fig. II-4) and a working floor for the next cut. The term "cut-and-fill" implies a definite characteristic sequence of operations: (1) Breaking a slice of ore from the stope back; (2) removing the broken ore; (3) introducing filling; then repeating the cycle.

Cut-and-fill stoping is applicable to mining firm ore enclosed within walls, one or both of which may be weak and heavy. Generally speaking, it is suitable for mining deposits too irregular for shrinkage stoping and deposits in which shrinkage could be employed were it not for the fact that the walls are too weak. Improved ore-handling and waste-spreading methods in recent years have extended the economic applicability of cut-and-fill stoping by comparison to shrinkage stoping and the method has the advantage of greater selectivity, better working conditions, and greater safety.

Timbered stopes: Timbered stopes are those in which timber is used systematically as a means of support. The most elaborate system of timbering is the square-set method, in which the walls and back of the excavating are supported by regular framed timbers forming a skeleton enclosing a series of contiguous, hollow, rectangular prisms in the space previously occupied by the ore and providing continuous lines of support in three directions at right angles to each other. (See fig. II-5.)

The ore is removed in small, rectangular blocks, usually just large enough to provide room for standing a set of timber. Ordinarily the stopes are mined in floors or horizontal panels one above the other, and the sets of each floor are framed into the sets of the preceeding floor. Timbered stoping is usually accompanied by filling; and often in heavy ground the sets are filled with waste promptly after they are installed, leaving only a small volume of the stope unfilled at any time. It has become accepted quite generally that, unless the ground is heavy enough to require filling for permanent support, the expense of timbering is not warranted and other methods should be employed.

Timbered stoping is adaptable to mining regular or irregular bodies, where the ore and/or walls are too weak to stand, even

FIGURE II-4
CUT-AND-FILL STOPE

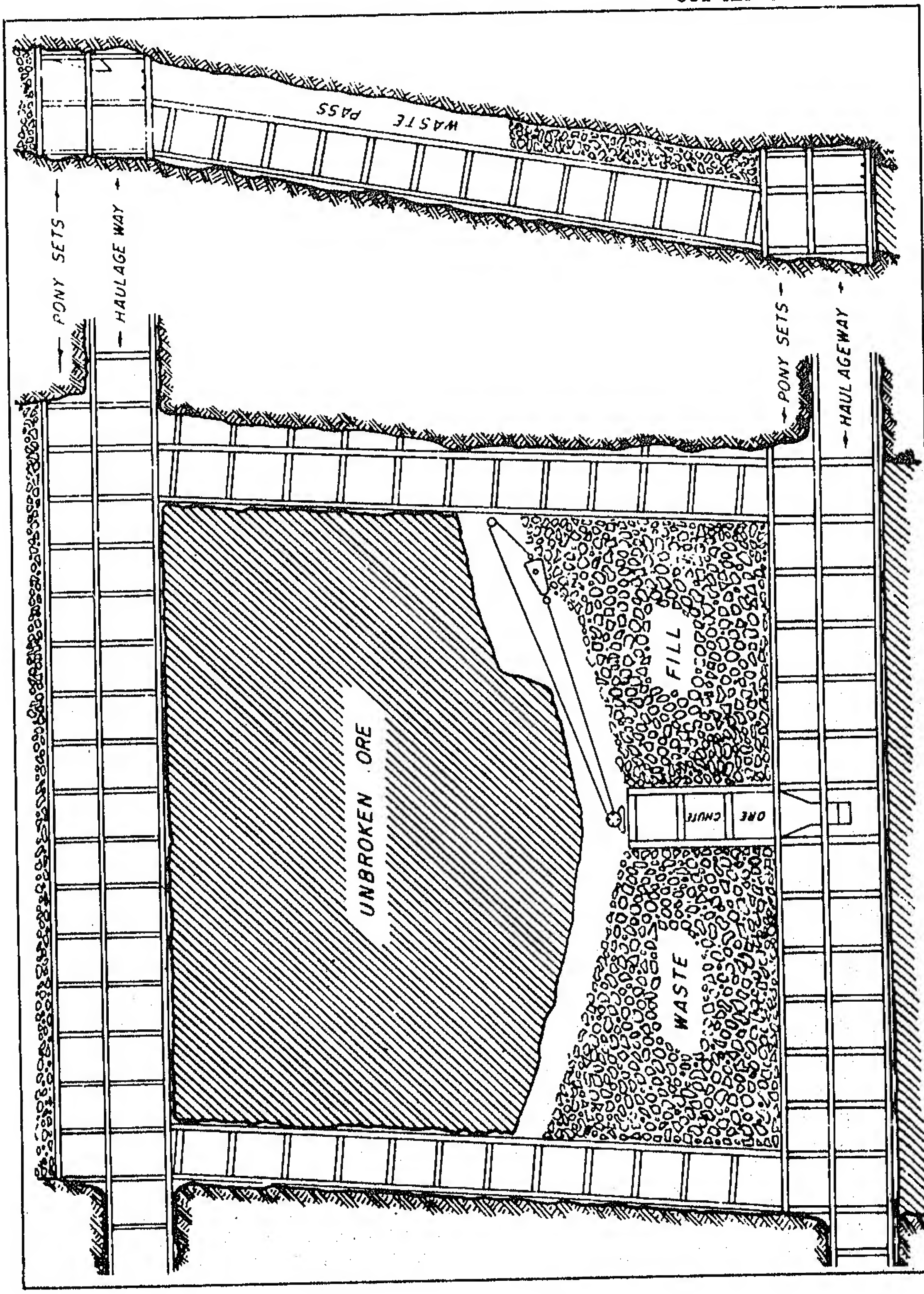
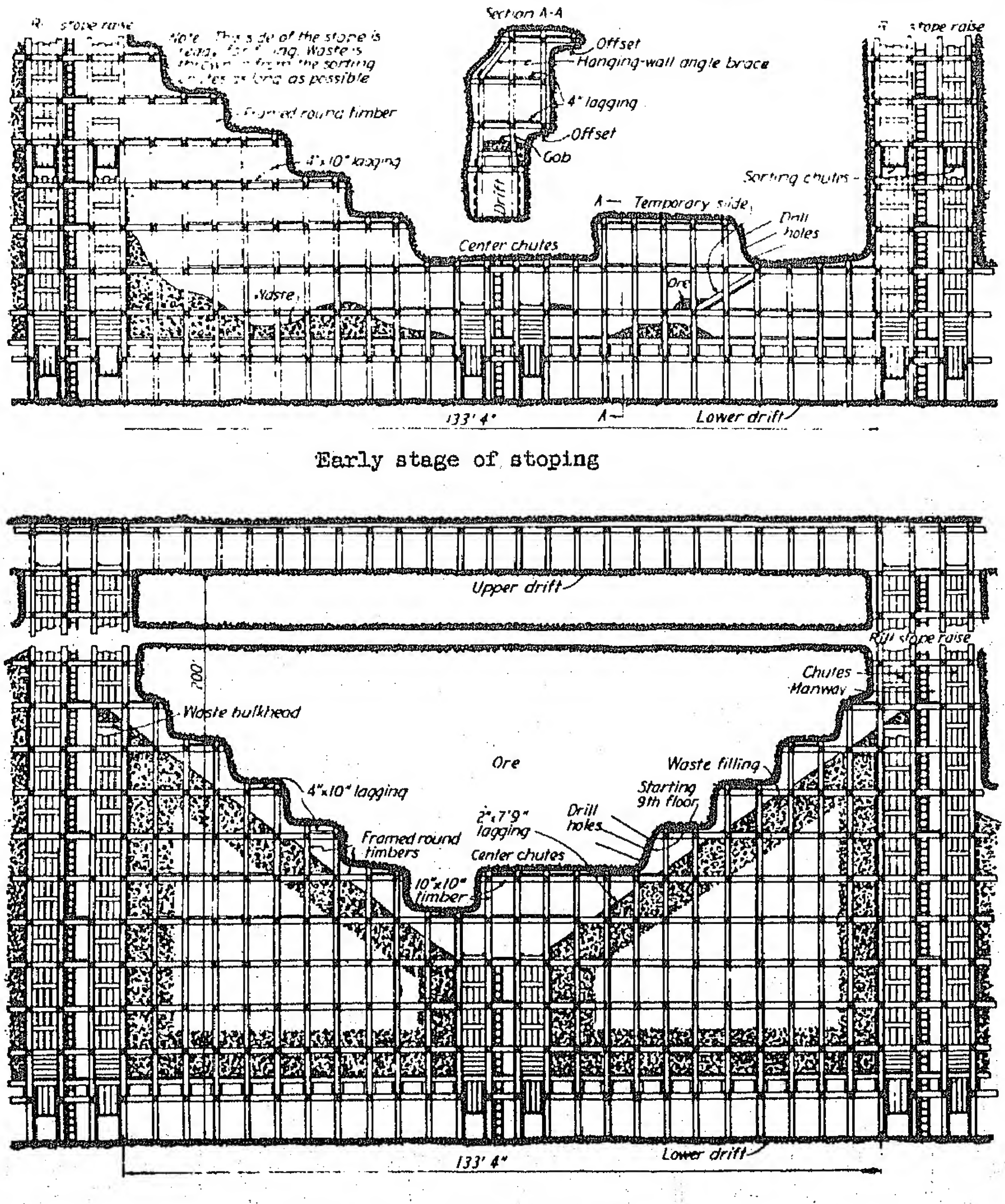


Figure II-4 Example of cut-and-fill stope
(From Bureau of Mines Info. Circ. 7560, March, 1950)

FIGURE II-5
SQUARE-SET STOPE



Early stage of stoping

Stope completely developed

Figure II-3 Square-set stope at Butte
(From Bureau of Mines Information Circular 6691, April 1933)

over short spans, for more than a brief time and where caving and subsidence of overlying rocks must be prevented. It is the most selective of the underground mining methods, hence is particularly suitable for mining rich, irregular ore bodies.

C. METALLURGICAL PROCESSES

The metallurgical extraction processes for the production of copper metal are based on the physical and chemical characteristics of the minerals, such as grain size, tenor, nature and content of valuable byproducts to be recovered, and type of impurities to be eliminated.

Low-grade oxidized-copper ores usually are treated directly by leaching, followed by precipitation (cementation) or electrolytic deposition of copper, and some sulfide and oxide ores are sufficiently high-grade for direct smelting, but the bulk of all native or sulfide copper ores are first subjected to a physical separation of minerals by an upgrading process known as concentration.

Sulfide concentrates and high-grade ores are treated in a smelter in a series of pyrometallurgical steps to produce an impure (blister) copper, which is subsequently refined by pyrometallurgical or electrolytic methods. Native copper concentrates are smelted, and the copper is fire-refined. In both smelting processes the gangue minerals and other valueless components are removed as a slag by the addition of suitable fluxes.

The distribution of the primary sources of copper from United States mines in 1950 is given in the following table.

MINE SOURCES OF COPPER PRODUCTION IN THE UNITED STATES, 1950

	Ore, short tons	Yield, percent	Copper, short tons	Percent of total copper
Concentrating ores, total	90,206,169	0.88	791,943	89.4
Native (Mich.)	4,386,474	0.58	25,608	2.9
Sulfides and oxides	85,819,695	0.89	766,335	86.5
Smelting ores	624,261	3.37	21,024	2.4
Leaching ores	3,755,362	0.88	32,922	3.7
Copper from precipitates	---	--	39,951	4.5
Total yield	93,961,531	0.90	885,840	

Byproducts may be recovered at one or more of the various steps of concentration, smelting, or refining. Thus in 1950, 89.4 percent of the mine production of copper in the United States was obtained from ores that required concentration, and the bulk of these were sulfides or mixed sulfide and oxide ores. Only 2.4 percent of the United States mine production of copper was derived from direct-smelting ores, 3.7 percent from leaching ores, and an additional 4.5 percent from the precipitation of mine waters and in-place leaching and precipitation. Elsewhere in the world, sulfide ores requiring concentration are also the major source of copper.

1. Concentration

Concentration is the term commonly applied to the process of effecting physical separations of two or more minerals. "Mineral dressing", "ore dressing", beneficiation, and particularly "milling" are also applied in the same connection, with slightly different connotations. The plants in which mineral dressing operations are conducted are known as mills or concentrators.

After liberation of the valuable minerals from the gangue, the separation of two or more minerals from each other is possible if they present critical differences in certain physical or chemical properties. The most important factors in the concentration and separation of copper minerals from the waste gangue are the chemical form, size, specific gravity, and surface characteristics of the several minerals in the ore.

The products of concentration are a concentrate, which contains the bulk of the valuable mineral, and a tailing which contains the gangue minerals. An intermediate product known as a middling may be re-treated in the plant for further recovery of valuable minerals before final rejection as a tailing.

Concentration is less costly than smelting. Furthermore, shipping costs to a distant smelter are reduced by producing a concentrate near the source of ore to avoid freight charges on the rejected waste.

Crushing and Grinding

The first step in concentration is to crush and grind the ore to such a degree as to liberate the valuable minerals from the gangue minerals, thus the grain size of the ore minerals is an important controlling factor. Hardness, tenacity, brittleness, and structure influence the cost of grinding, and the relative degree of "sliming" or production of extreme fines of each of the minerals in an ore. Crushing and grinding are done only to the size necessary to liberate copper minerals from the gangue since sliming results in recovery losses. Few domestic ores contain enough copper minerals of sufficient size to permit concentration of coarse fragments, hence most crushing plants for copper ores obtain size reduction in three steps: Coarse crushing from run-of-mine sizes to 2 inches diameter or finer, usually performed in gyratory or jaw crushers; intermediate crushing in jaw, gyratory, or cone crushers or by rolls to 1/4 inch or larger - at this state auxiliary concentration methods for waste elimination may be employed; and fine grinding in ball mills or rod mills. Operating practices vary widely with the tonnage treated and with the varied character of the ores from different districts.

Figure II-6 is a diagrammatic flow sheet of a typical two-stage primary crushing plant, illustrating the major units of equipment and their functions. Gyratory and jaw crushers have been built to receive rock as large as 60 inches in diameter. The maximum degree of diameter reduction in a single unit is preferably not over 4:1, but may be as high as 6:1, hence primary crushing of large-size feed requires at least two stages to reduce the particle size to 2 inches or less.

Fine grinding in ball or rod mills usually requires a feed less than 2 inches in maximum particle size. Although it is possible to grind 2-inch-diameter feed to about 48-mesh (0.012-inch diameter) in a single stage, grinding is usually cheaper if done in two or three stages when tonnage to be ground is great enough to justify more than one grinding unit, and a high degree of communition is required. Ball mills commonly are used for fine grinding and rod mills for coarse grinding.

Screens and classifiers play an important part in mineral dressing. Screens and grizzlies are used to bypass material already fine enough or to return oversize for recrushing. They are most efficient in size ranges above 10-mesh (0.065-inch diameter), below which size various types of classifiers are preferred in most metallic mineral concentrators.

Classifiers are capable of separating finer sizes of materials into groups according to their settling rate in a fluid, usually water, or air. Of two particles of equal density but of unequal size, the larger will settle faster, and of two particles of equal size but differing in density, the higher density will settle faster. The chief applications of classification in modern ore-milling practice are in closed circuit with grinding mills to return oversize for regrinding, and as thickeners for removing part of the water from fine concentrates, or to recover part of the water from fine tailings. Classification is also used to prepare table feed, in gravity concentration but tabling plays a comparatively minor part in modern milling practices.

The more important accessories to ore concentrators are conveyors, bins, pumps, feeders and filters. Some of these units and their functions are illustrated in figures II-6 and II-7.

<u>Gravity Concentration</u>

Differences in the specific gravity of minerals form the basis of a gravity concentration process. The principal types of gravity concentration are: (1) <u>jigging</u>, dependent upon differences in the settling rate of minerals as they are carried horizontally by water

Mined ore must be crushed to a small size to free the valuable minerals so that they can be removed. Usually, the ore feeds over a grizzly which by-passes the fines and sends the oversize to a jaw crusher. From here the fine ore passes through a screen and is taken directly to storage bins by a conveyor. The coarse ore goes through a gyratory or cone crusher before going to the storage bins. Each bin is equipped with a feeder which conveys the ore it contains to the mill.

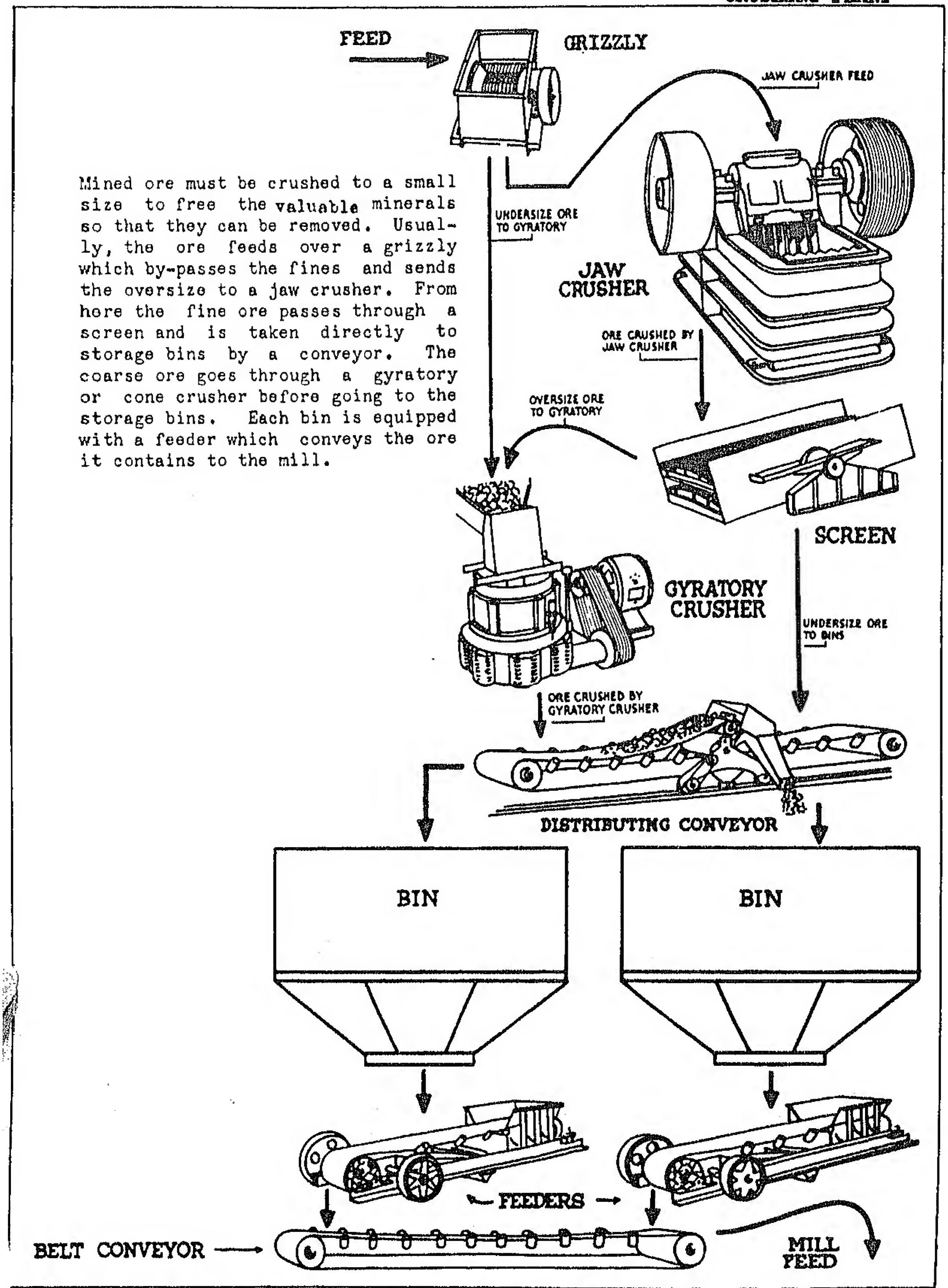


Figure II-6 Major equipment and flow sheet in a typical crushing plant

flow over a screen bed and subjected to a vertical pulsating action; (2) heavy-media separation methods, which are dependent upon the use of a fluid of sufficiently high gravity or a suspension of a heavy mineral in water such as galena or ferrosilicon, to cause the gangue minerals to float and allow the ore minerals to settle; (3) tabling dependent on the differences in the speed of travel of minerals of different sizes and gravity as they are caused to flow in a stream of water or air transversely across an inclined riffled surface that is subjected to a longitudinal reciprocating motion.

In connection with mineral dressing, jigging and heavy-media concentration processes are applicable only to the separation of minerals of differing specific gravity that are partly or wholly liberated from each other by crushing to approximately 48-mesh (0.012-inch diameter) or larger.3/ The particle size and gravity-differential limitations of each method overlap substantially. The maximum particle size that can be treated by either method is limited more by considerations of liberation than by process technology. Jigging has been used for ore-gangue separations on sizes up to 3 inches in diameter and presents greater technologic difficulties in the design and operation of equipment for larger size than heavy-media separation. In fact, heavy media could conceivably be used as a preconcentration method on run-of-mine rock from nonselective mining after crushing, say, to 8 inches in diameter. In present practice however, the maximum size being treated is 2 or 3 inches, as most ores require crushing at least that fine in order to free sufficiently the various mineral from the gangue.

The minimum size that can be treated by either jigging or heavy-media separation is about 48-mesh if there is a considerable gravity difference between the minerals to be separated and about 10-mesh (0.065-inch diameter) where the gravity difference is small.

Jigging is effective if the minerals to be separated have a specific gravity difference of 0.5 or more although, if closely sized fractions of the feed are treated on separate jigs of suitable design, the gravity differential may be as little as 0.25.

Neither jigging nor heavy-media separation is extensively used in copper-ore concentration at present, because most ores require finer grinding for mineral liberation than the minimum economic size limit of gravity methods. Most copper sulfide minerals are recovered

---

3/ Modifications of both processes have been applied to nonmineral separations, as in grading certain agriculturial and food products.

by flotation mills: A pilot table is often used to help the control of the losses in the flotation circuit.

Flotation

Flotation is a process of wet concentration in which air bubbles are used to float one kind of particle from a mixture of two or more kinds of finely divided materials suspended in water. Certain minerals may be preferentially oiled by organic reagents in the presence of water so that they will adhere to air bubbles and float to the surface of the pulp. Other minerals, usually the gangue, are unaffected and remain suspended in the water.

Although considerable progress has been made in recent years in applying flotation to nonsulfide minerals, the flotation of the oxide, carbonate, and silicate minerals of copper is still unsatisfactory due to low recoveries, low-grade concentrates, high cost of reagents, and the difficulties of operating control.

The reagents that selectively coat the valuable mineral particles are known as collectors. In sulfide flotation these are usually xanthates or other chain hydrocarbons. Supplementing the action of the collectors is a group of reagents known as frothers whose function (in conjunction with agitation and aeration) is to create a myriad of small bubbles in the pulp. The bubbles attach themselves to the properly conditioned valuable mineral and rise to the surface to form a froth which may overflow or be removed mechanically. Frothers in sulfide flotation are usually amyl alcohol, pine oil, or ring-carbon compounds, such as cresylic acid.

When two or more minerals of the easier floating type are all floated together to form one concentrate, the process is known as bulk flotation. Differential flotation is the term used to describe an operation in which one or more sulfide minerals are depressed during flotation of one or more other minerals, or where several different sulfides are floated successively into separate products. The usual differential separations are copper sulfides from pyrite, galena from sphalerite, and sphalerite from pyrite. Any one of the sulfides may be rendered more floatable than the others by the correct surface modifications, and such modifications are obtained by the use of a class of reagents known as conditioning agents. Under this class are grouped the depressing, activating and dispersing agents, the pH regulators and the cleaning agents.

In general, sulfide-mineral particles coarser than 35-mesh (0.016-inch diameter) cannot be effectively recovered by flotation; consequently, an ore that is to be floated must first be ground

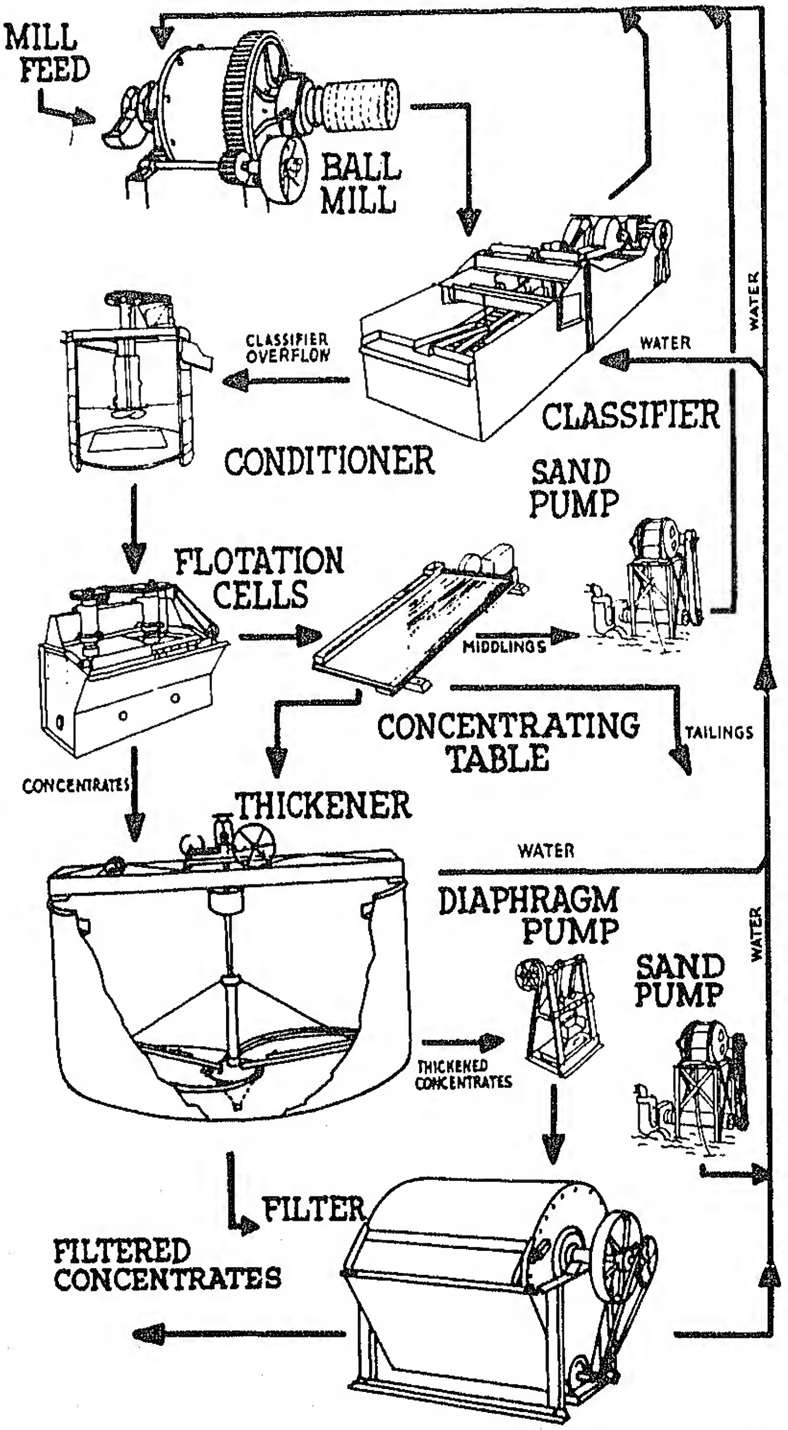
fine enough so that all, or substantially all, the desired mineral is smaller than this limiting size. This is aside from considerations of liberation, which may require even finer grinding.

To obtain a good recovery and a high-grade concentrate, several stages of flotation concentration are required. This is conveniently accomplished by employing a number of successive agitating chambers or cells, so that the tailings from the first cell pass progressively from one cell to the next as the original ore becomes impoverished in metal content. The concentrate from the first group of cells treating low-grade ores is seldom rich enough for a final product and is known as a rougher concentrate, which is re-treated or "cleaned" in one or more stages to produce a cleaner concentrate. In some cases, the rougher tailings may require regrinding for further liberation of minerals and in other cases may be floated directly to produce a low-grade scavenger concentrate, which is often returned to the first rougher cell.

A typical flow sheet of a single-product ball-milling and flotation plant is given in figure 7, p. 32. A classifier is shown in closed circuit with a ball mill to return oversize for regrinding. A heavy circulating load may be used to avoid overgrinding and to obtain higher output of finished product from an installation of a given size. The classifier overflow is fed to a flotation machine, to which various reagents are added by mechanical feeders. The flotation tailings are passed over a concentrating table, which is used primarily as a visual check on the efficiency of the grinding and flotation circuit, though some concentrates too coarse for efficient flotation are recovered. The combined concentrates are thickened, them dewatered in a small vacuum filter. The plant tailings are conveyed by gravity flow or pumped to a settling pond. Where more complete water recovery is required, the tailings may be thickened and filtered, then mechanically conveyed to a tailing pile.

Figure II-8 is the flow sheet of the Morenci concentrator of the Phelps-Dodge Corp. of Morenci, Ariz. This is one of the largest and most modern installations in the world. The plant has a capacity of 45,000 tons of ore per day, carrying about 1.00 percent copper, the chief ore minerals being chalcocite and chalcopyrite. The crushing plant is capable of handling more than 4,500 tons an hour. Crushing is carried out in three stages to 3/8" and further reduction by single-stage grinding before flotation with regrinding of rougher concentrates for final cleaning. The scavenger froth and the "cleaner" tailing are reground in a separate circuit. Recovery is 87 percent of the total copper, with a ratio of concentration of 30:1. The concentrates sent to the smelter carry about 27 percent copper and the tailing 0.12 percent copper.

FIGURE II-7
FLOTATION PLANT



MILLING

Ore is milled to free the precious materials from as much of the barren rock or gangue as possible.

In the flotation plant the **ore** is first ground in a ball mill, the fines feeding to a conditioner where flotation reagents are added. The pulp then flows to flotation cells which separate the values. Tailings from the flotation cell are sometimes passed over a concentrating table to trap valuable minerals that were too big to be flotated.

Flotation concentrates pass through a thickener and then through a filter and sometimes through a dryer to remove as much as possible of the excess moisture.

Figure II-7 Major equipment and flow sheet in a typical flotation plant

FIGURE II-8
COPPER CONCENTRATOR

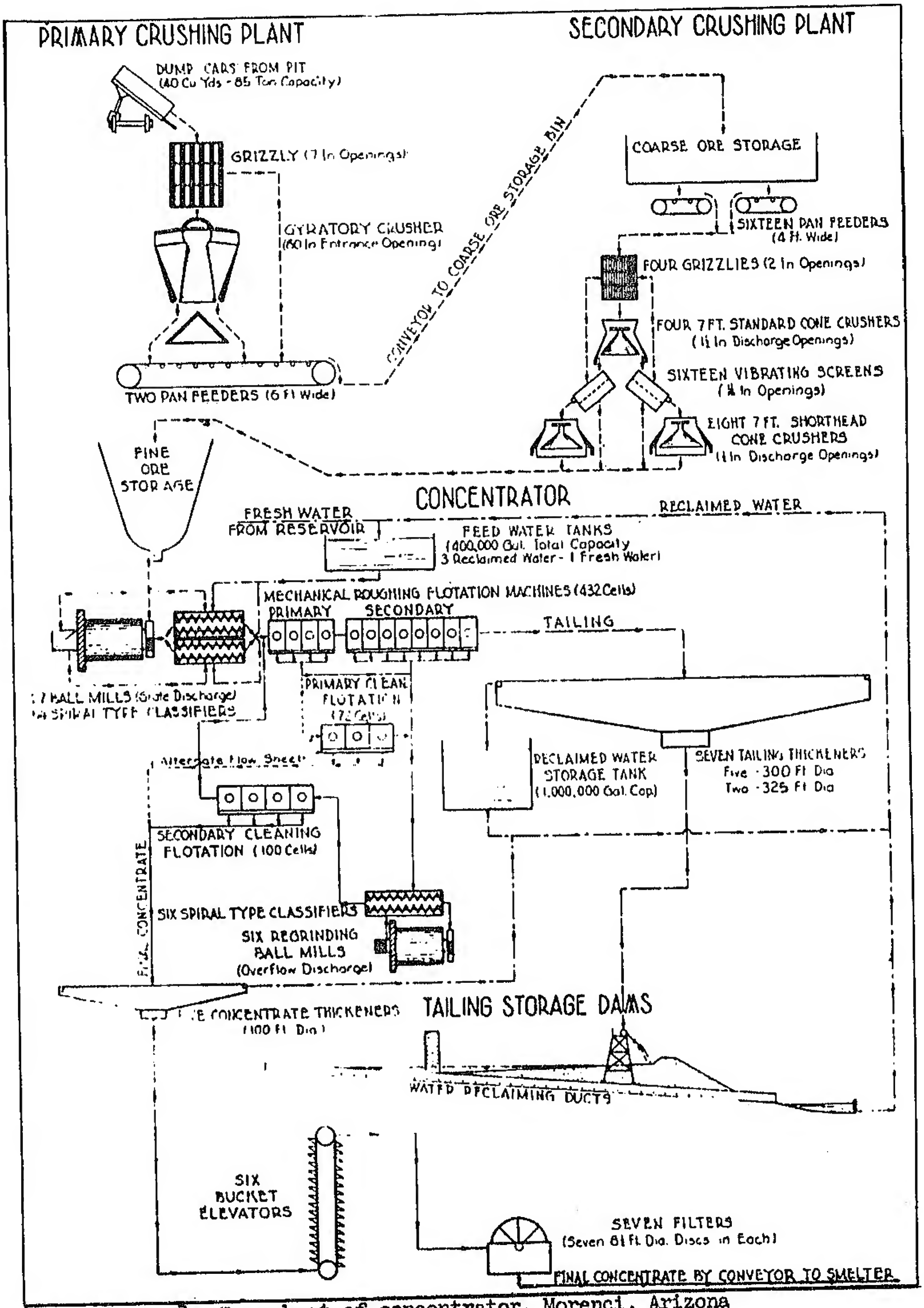


Figure II-8 Flow sheet of concentrator, Morenci, Arizona

With "mixed" ores in which both sulfides and oxidized minerals occur, the treatment depends upon the relative proportions of the two kinds of minerals. If the sulfide minerals predominate, flotation is used, employing reagents that favor the flotation of the oxidized minerals. With such treatment, it is usually possible to recover over 90 percent of the sulfide copper and some 50 to 70 percent of the remainder. When oxidized minerals predominate, the copper is usually recovered by leaching with sulfuric acid.

When there are almost equal amounts of sulfide and oxidized ores, combinations of leaching and flotation may be employed. The Lark Mill at Bingham, Utah, uses another modification. The oxidized minerals are dissolved in sulfuric acid, then, without filtering, the copper is precipitated by agitation with scrap iron. The pulp containing the sulfide copper, and the precipitated copper is then floated, recovering both materials.

2. Smelting

Since most world copper production is extracted from low-grade sulfide ores that require concentration, the dominant metallurgical processes for recovering copper metal are adapted to treating fine-grained sulfide concentrates. This involves three major steps: Roasting, reverberatory smelting, and converting. Roasting removes part of the sulfur and produces a calcine that is smelted in a reverberatory furnace, with some additional elimination of sulfur to produce a copper matte that contains most of the copper and some of the iron as sulfides and collects the gold, silver, and certain other metals. The question of removal of sulfur by calcining or by the converting operation is a matter of economics and conservation of copper. The older practice used more calciners and charged more iron oxides to the matte furnace. More modern practice tends to eliminate the calciners and to produce a lower grade matte and consequent higher recovery of the copper, leaving more sulfur and iron to be removed in the converter. Impurities are removed in the reverberatory furnace as a slag with the aid of fluxes to form fusible compounds. The matte is then blown with air in a converter to remove the sulfur as sulfur dioxide and to oxidize the iron so that it may form a slag with the siliceous flux, which often contains precious metals. The product of the converter is blister copper.

The older process of direct smelting of oxide ores, usually over 25 percent copper, to produce black copper is obsolete, except in isolated localities, for want of suitable ores and because of high slag losses. Since concentration of low-grade oxide ores results in high tailings losses, direct leaching of such ores is common practice. The concentrates from mixed oxide and sulfide ores may contain 60 percent or more of the oxide copper content, but

the tailings often contain enough copper to warrant leaching as an additional step.

Most copper smelters accept for custom treatment ores that are valuable chiefly for their gold and silver content in conjunction with desirable fluxing characteristics, and the copper ores themselves frequently contain precious metals. Certain other metals, such as arsenic, antimony, lead, selenium, and tellurium, accumulate in the blister copper. These must be removed and may be recoverable as marketable byproducts. Other components, notably sulfur and oxygen, must be removed to produce a refined copper for the market. The customary refining process for blister copper involves a preliminary fire-refining stage, followed by electrolysis and a final fire refining. The product is known as electrolytic copper.

The treatment of the native copper ores of the Lake Superior district in Michigan departs from the general pattern. The ore is concentrated, smelted, and the copper is fire-refined. The content of precious metals is too low to warrant the expense of electrolytic refining, and the resulting copper is sold as Lake copper.

Figure II-9 shows generalized flow sheets for reverberatory or blast-furnace smelting, with typical partial analyses of materials at various stages. Details of practice vary widely with conditions at each plant.

Roasting

Roasting is a pyrometallurgical process that consists of heating an ore or concentrate in an oxidizing atmosphere to effect oxidation reactions. The end reaction of roasting metal sulfides is given in the following equation:

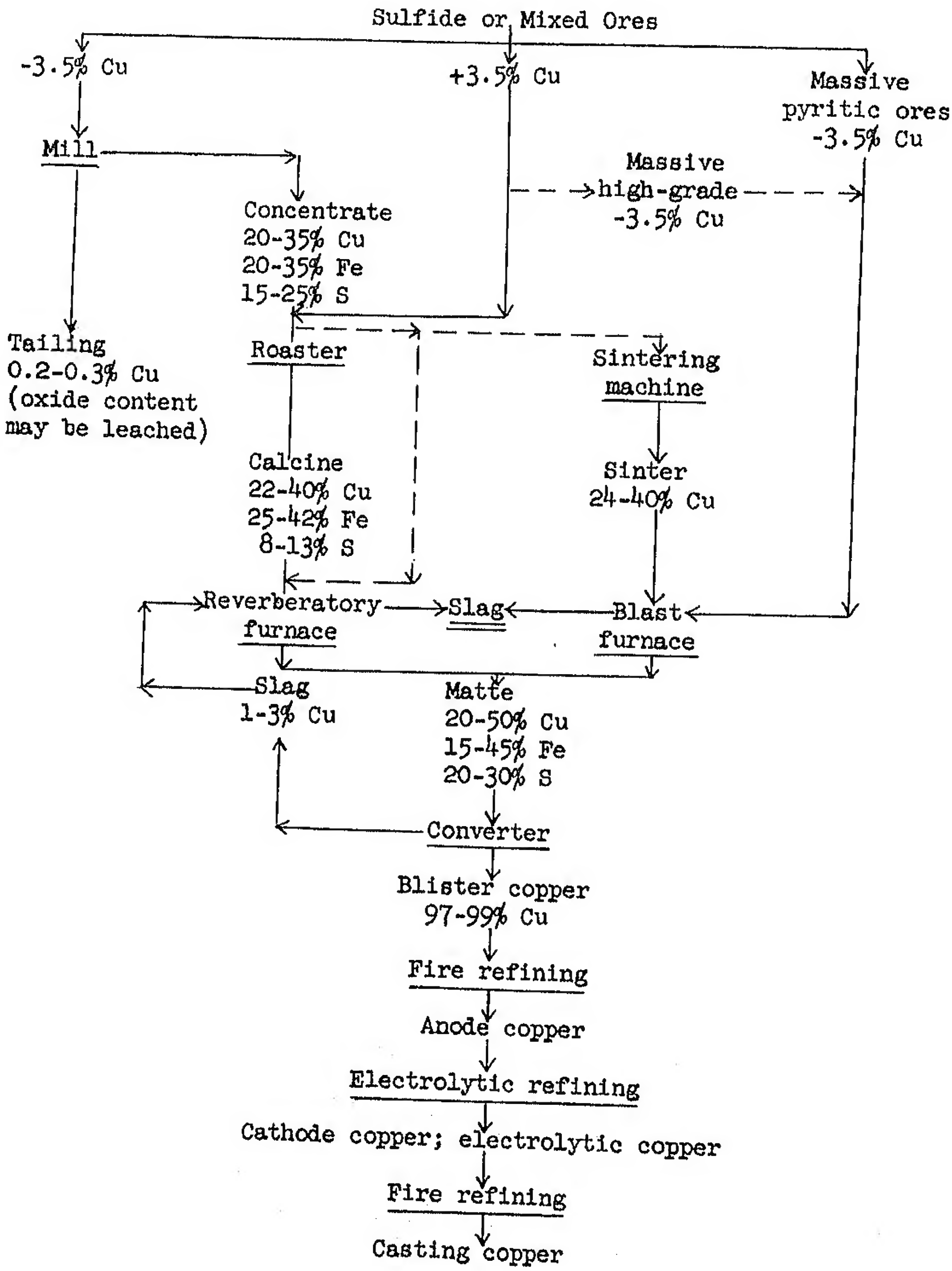
METAL SULFIDE	OXYGEN (air)	(HEAT)	=	METAL OXIDE	SULFUR DIOXIDE

Copper and iron sulfides are among the easiest of the common metallic sulfides to convert to oxides, with minimum production of sulfates and other intermediates; furthermore, mixtures of copper and iron sulfides relatively free from inert compounds are readily ignited and will roast autogenously (that is, independently of other reagents) to 6 to 8 percent sulfur without further application of heat, the combustion of sulfur and the oxidation of iron and copper being strongly exothermic (that is, accompanied by the evolution of heat) reactions.

In copper smelting, a complete or "dead" roast is not desired, as enough sulfur must be left in the calcines to form with the

Figure II-9

GENERAL FLOW SHEET OF COPPER SULFIDE ORE TREATMENT



copper and part of the iron in the ore a matte of suitable grade for subsequent converting to blister copper. The copper blast furnace is capable of utilizing some or all of the excess sulfur as a fuel, but the reverberatory furnace eliminates only a small part of the sulfur content of the feed, hence the residual sulfur in the calcines depends upon the smelting process used as well as upon the initial content of copper, iron, and sulfur and the desired grade of matte.

The concentrates produced from some ores may be controlled at the concentrator to avoid the need of roasting before reverberatory smelting. This is the practice at the smelter of the Phelps-Dodge Corp. at Morenci, Ariz. This alternative is shown in figure 8 as a bypass of concentrates directly to the reverberatory furnace. Representative analyses of the products and intermediates at Morenci are given in the following table, from which it can be noted that the copper, iron, and sulfur content of the concentrates is such that only the normal sulfur loss in the reverberatories is needed to produce a matte of suitable grade for converting. Furthermore, the gangue, aided by the converter slag as a source of iron oxide, is essentially self-fluxing.

Modern roasting practice has developed along two principal lines: Hearth roasting and blast roasting. Various types of multiple-hearth roasters are preferred in copper smelters to roast sulfide feed for reverberatory smelting. Blast roasting, usually in the Dwight-Lloyd machine, is used at some copper blast-furnace plants to sinter and partly roast fine-grained ore and concentrates and is standard equipment in lead smelters for dead roasting and sintering.

Hearth roasting in the multiple-hearth furnace is by far the most prevalent method for roasting copper ores and concentrates. The first single-hearth roasters were hand-rabbled, and later a number of different types of mechanically rabbled roasting furnaces were developed. These have been gradually improved mechanically, and today the standard furnace for roasting of copper concentrates and ores is the cylindrical, multiple-hearth, mechanically rabbled furnace, as shown in figure 10.

The design of these furnaces permits constant and accurate control of the material in process at all points. Thermocouples can be installed in the rabble arms to permit the operator to read the temperatures on each hearth. Burners are set in the side walls to be used when the ore or concentrate does not contain enough sulfur to be self-roasting. The operator can regulate the speed, air supply, and temperature in such a way to obtain the maximum roasting efficiency for the material being treated.

ANALYSIS OF MORENCI SMELTER PRODUCTS
(Ore 1.0 percent copper)

Analysis of:	Au, oz.	Ag, oz.	Cu, %	SiO_2, %	Al_2O_3, %	Fe, %	CaO, %	S, %	FeO, %	H_2O, %	Fe_3O_4, %	O_2, %
Concentrate smelted	.0172	1.132	27.24	5.4	1.6	26.2	.1	37.6				
Reverberatory solid charge	.015	1.00	24.10	10.4	1.8	23.1	3.2	32.8		9.04		
Reverberatory slag	.00029	.0156	.44	37.3	6.9		4.7	1.1	46.0			
Reverberatory matte	.021	1.393	35.72			33.9		26.3			4.3	
Converter flux	.0013	.063	1.43	65.0	10.6	10.2	1.5	2.0				
Converter slag	.0013	.0834	2.93	26.2	4.9	45.5	.7	1.5			18.0	
Anode copper	.064	4.13	99.76					.0018				.098

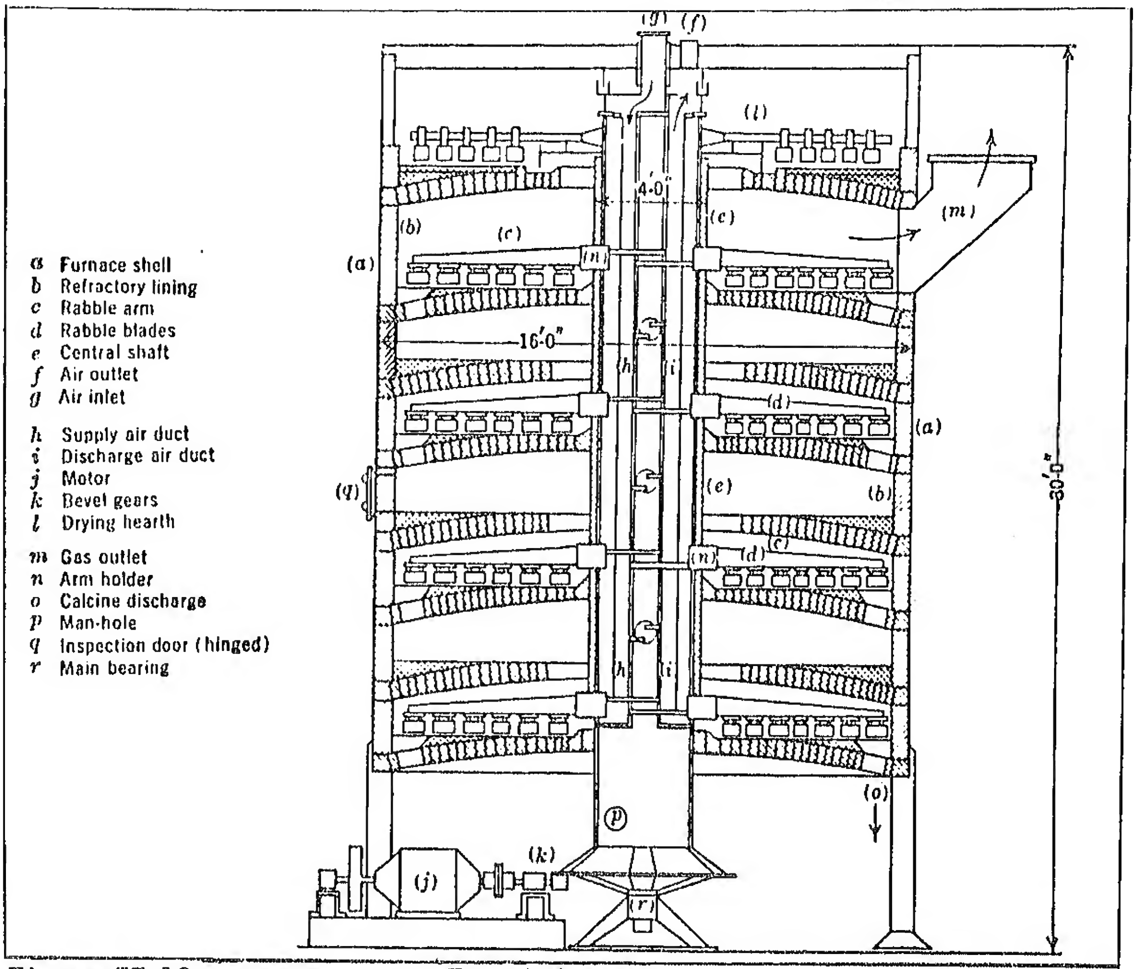


Figure II-10 Turret Roaster

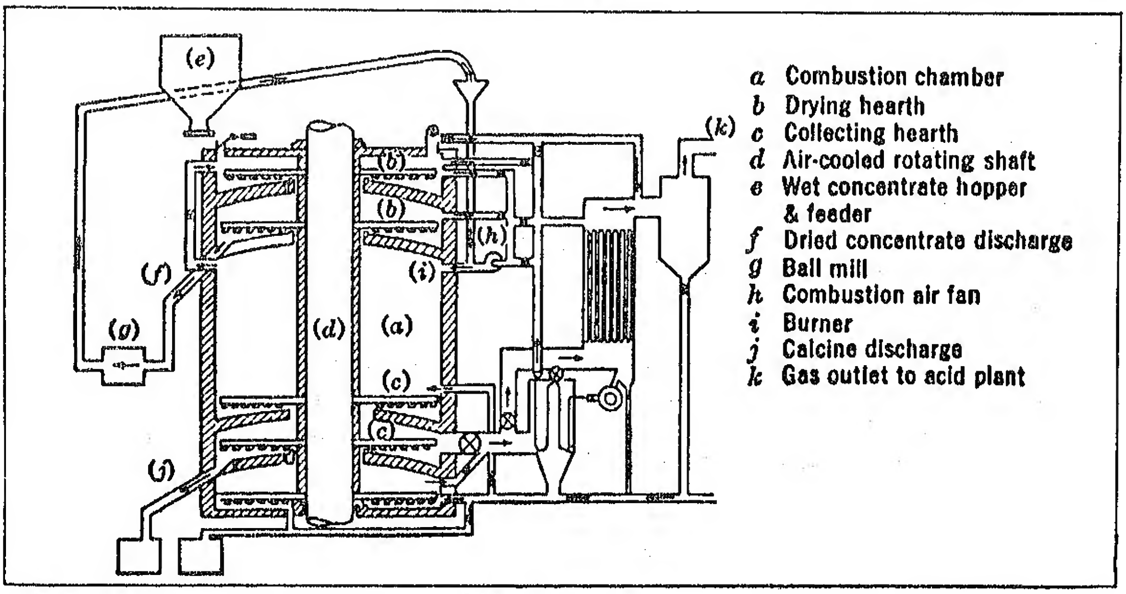


Figure II-11 Flash Roaster

In operation of the multiple-hearth furnace, most of the roasting takes place while the ore is dropping from hearth to hearth, opposing the uprushing current of hot air. As the sulfide particles fall through this air, their entire surface is exposed to the action of the oxygen in the air and consequently oxidizes rapidly. In recent years, a process known as flash or suspension roasting has been developed for which the roasting apparatus is designed so that all the roasting is done while the particles are falling through the heated air. Flash roasting requires that the ore or concentrate be very finely divided. The relation of flash roasting to hearth roasting is about the same as the relation of powdered-coal burning with burning of lump coal on a grate. Although a flash roaster may have a much greater capacity and require less fuel than a rabbled furnace, dust losses are likely to be high. Figure 11 illustrates an installation of this type at the smelter of the Consolidated Mining & Smelting Co., Trail, British Columbia. Three hearths of an eight-hearth turret roaster were removed, leaving two drying hearths at the top and three hearths at the bottom for collecting and finishing the calcines. The dried feed is ball-milled, then injected into the open chamber of the furnace with enough air for combustion. About 60 percent of the particles settle on the collecting hearth and the remainder is collected in cyclone dust collectors.

Blast roasting: The essential features of the Dwight-Lloyd sintering machine are illustrated figures 12 and 13. The machine comprises a structural steel framework supporting a closed track around which travel a series of small grate-bottom cars or pallets for carrying the charge; a driving mechanism; suction boxes beneath the upper pallet track section connected to an exhaust fan for drawing air through the bed; a feed hopper; and an igniter for starting combustion of the fuel in the charge. The charge may consist exclusively of sulfides; or, if the sulfur content is too low for autogenous roasting, coke breeze or coal may be added, and in some cases fluxes are required to prepare a sinter that is more easily smelted in the blast furnace.

The charge is fed to the mixer, where it is moistened, mixed, and worked into an air-permeable condition, then it passes through the distributing device, which delivers it evenly across the full width of the pallet behind the feed hopper. As the pallet moves from under the feed hopper the charge passes under the igniter at the front end of the suction box. The igniter may be fired with any suitable fuel; its purpose is to project an intense flame on a small area of the upper surface of the bed and kindle the sulfur or other fuel in the charge. After passing the igniter, the charge must cross the suction boxes, where sintering takes place, and is finally discharged as finished sinter cakes. The Dwight-Lloyd

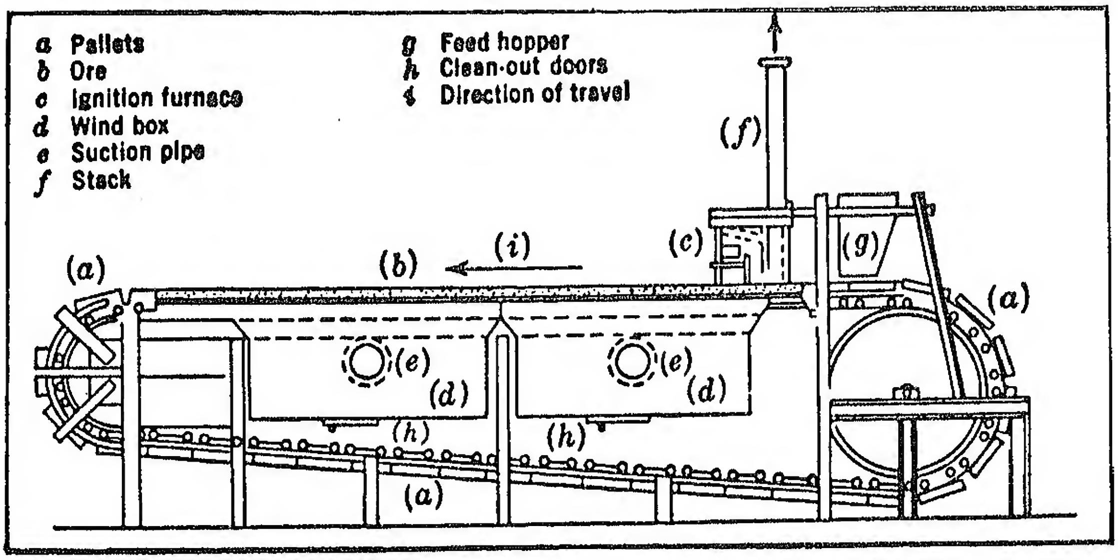


Figure II-12 Dwight-Lloyd sintering machine

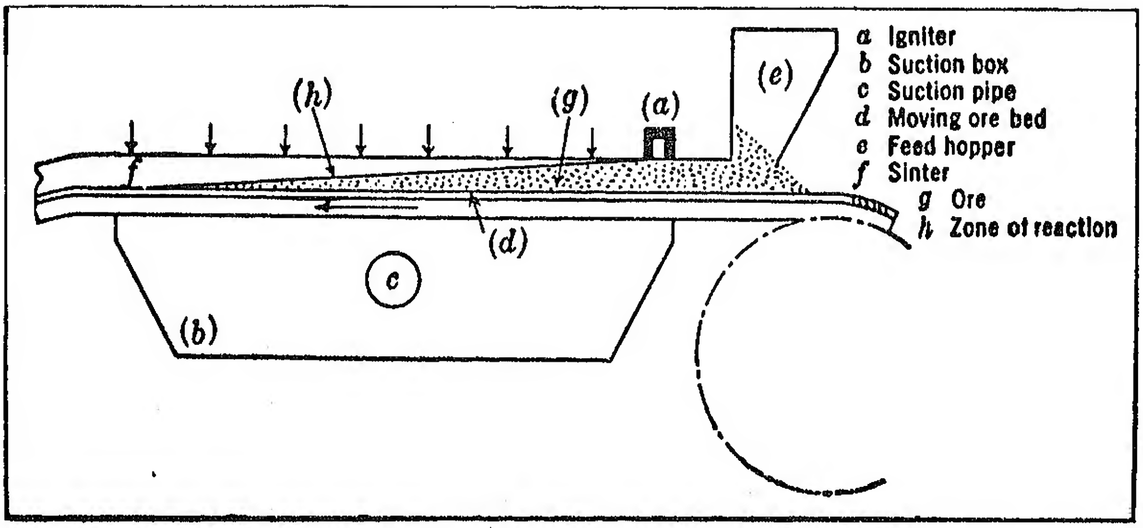


Figure II-13 Longitudinal section through sinter bed
Dwight-Lloyd sintering machine

process produces a sinter that makes an excellent feed for the blast furnace; it is free from fines and dust and strong enough to support the weight of the charge; it is porous and hence readily permeable to gases; and the material is prefused so that it smelts more easily.

Fluosolid roasting: Fluosolid roasting, or "fluosolids", a new roasting technique adapted from the "suspended-solids technique" of the petroleum-refining industry, has recently been introduced to the copper industry for roasting copper ores. The technique consists of partial suspension of solid particles in a gaseous stream, producing an effect almost exactly comparable to the sorting action in the column of hindered-settling classifier. With "fluosolids", however, the gas velocities are such that size segregation of properly prepared feed will not occur. The entire fluidized bed, which is in turbulent motion like a boiling liquid, is substantially uniform throughout.

Copper sulfides can be sulfatized with "fluosolids", making the copper water or weak-acid soluble. Calcines have consistently been obtained in which as much as 90 percent of the copper is water-soluble, with an additional 5 to 9 percent soluble in 5-percent sulfuric acid. At the same time, close temperature control allows only 1 to 1.5 percent of the iron to go into solution. Two possible applications are suggested: First, copper sulfide concentrate could be roasted, then leached, then treated electrolytically, bypassing the smelter entirely. Second, copper-gold ores, which could not be cyanided because of the copper, could be given a sulfating roast, followed by a water or weak-acid leach for the copper and cyanidation for the gold. This effect could also be used to produce blast-furnace feed from cupriferous pyrite.

Reverberatory furnace smelting: A reverberatory furnace (see fig. 14) is a long, shallow furnace consisting of a hearth, side and end walls, and a roof. Reverberatory furnaces are of varying size, with dimensions usually of the order of 110 feet long, 25 feet wide, and 10 feet high. Furnace capacities also vary considerably - from as low as 50 tons of charge to as high as 1,500 tons (or higher) daily. The furnace is heated by means of burners placed in one end wall, and the products of combustion escape at the other end. A long-flame fuel is used - gas, fuel oil, or pulverized coal - and the flame extends over a large part of the hearth. The material on the hearth is heated by radiation from the flame. There is ordinarily no extensive reaction between the gases in the furnace atmosphere and the charge on the hearth; it is possible to get some oxidation of the charge by using a large excess of air for the combustion, but this wastes heat and is seldom practiced.

FIGURE II-14
COPPER REVERBERATORY

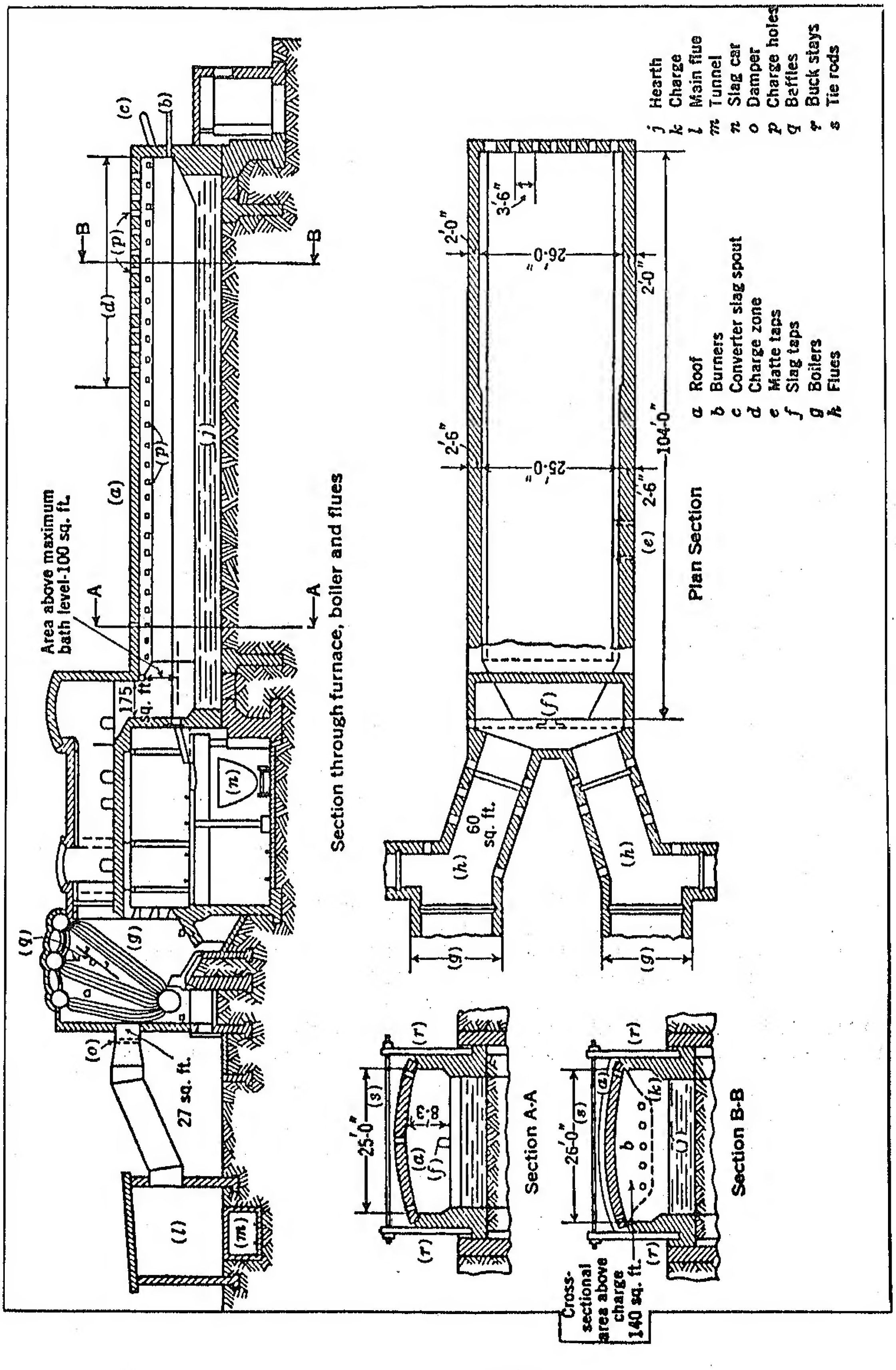


Figure II-14 Reverberatory matting furnace for copper ores

As a rule, the principal chemical reactions that take place in the charge of a reverberatory furnace are reactions between various constituents of the charge itself to form a matte and a slag. Reverberatory slags are essentially mixtures of silicates and aluminosilicates of iron and lime. In many cases, the charge derived from copper bearing materials is self-fluxing; but in some cases barren fluxes, such as limestone, are added to obtain a suitable slag. A typical slag analysis is given on page 38.

The foundation of the furnace is made of concrete or poured slag. The hearth is 24 to 30 inches deep and lined with a refractory material. Silica (acid) is generally used, but magnesite (basic) is preferred at some smelters. The side walls are usually of silica brick, but if a magnesite hearth is used a layer of chrome brick is laid to separate the acid side walls from the basic bottom.

At points along one side, one about 25 and the other about 45 feet from the bridge or burner end, two matte tap holes are provided on a level with the bottom of the hearth. At or near the flue end of the furnace, at a higher level, a skimming hole is provided for the slag. Water jackets are used at some plants for cooling the hearth, ports, and other special parts. On a level with the top of the hearth and along both sides of the furnace are small openings with tightly fitting sliding doors. Through these the operators may watch the changes in the contents of the furnace, rabbling the charge if it is found necessary and making temporary repairs to the bottom and the side walls.

The general shape of the roof must reflect or "reverberate" the flame downward onto the charge on the hearth. It is customary to "dome" the furnace section at the combustion end. Doming consists of raising the roof about 2 feet higher than the normal arch to provide sufficient combustion space. It cannot be done by raising the arch across its entire width because of the nature of the fine - hot calcines which have such a flat angle of repose that any great increase in the elevation of the drop holes would result in blocking the center channel of the furnace. The efficiency of the furnace depends largely upon the contour of the roof. It varies in thickness from 15 to 20 inches and may be either the sprung arch or supported type. There is a distinct tendency toward the use of magnesite roofs of the suspended type, but silica brick is still in general use. The experience of one of the large smelters (3) is significant. There it has been found

that the life of a magnesite suspended arch is many times that of the sprung silica type. Furthermore, the furnace campaign can be prolonged almost indefinitely by "hot patching", that is, replacing burned-out brick without reducing the firing rate or interfering with normal furnace operation. The lost time for repairs per furnace per year has been reduced from 12.6 to 3.7 furnace days.

In modern practice, because calcines are so frequently used, a hot reverberatory charge is required. The old method of charging was to drop large amounts of cold ore and flux onto the hearth through hoppers along the center line of the furnace near the fire-bridge end. These large charges lowered the temperature of the furnace and interfered with the best working of the furnace. In recent years, continuous side charging has been commonly employed. In this method the hot, calcined ore and flux from the roaster are introduced into the furnace through water-cooled pipes, which are distributed along both sides of the first 25 to 30 feet of the firing end. Operating in these holes are powerful screw feeders driven by air or electric motors. Material is supplied to these screws by feed bins located just below the feed floor and along the sides of the furnace. Powerful plungers are frequently used instead of the screw feeders for the introduction of the ore. Converter slag is added, while molten, through the ports. Continuous side charging reduces dust loss materially, protects the side walls of the furnace from the mechanical wear of the charge, as well as from the corrosive action of the slag and matte, and generally prolongs the life of the roof by cutting down spalling and cracking caused by rapid temperature changes.

The products of the reverberatory furnace are matte, slab, speiss, and flue dust. With respect to mattes, a much higher ratio of concentration and, consequently, higher-grade matte may be produced than in the blast furnace.

A recent survey has shown that half the domestic plants use natural gas; most of the remainder, pulverized coal. The amount of coal consumed in a large furnace ranges from 275 to 400 pounds per ton of charge, that of fuel oil from 0.5 to 1.5 barrels per ton of charge. There is considerable loss of heat in the waste gases from these furnaces, which leave at about 1,200° C.; consequently, waste-heat boilers are frequently provided, whereby as much as 55 (average 40) percent of this heat can be utilized for the generation of power.

Blast-furnace smelting: The copper blast furnace is used mainly for smelting coarse ores over 1 inch in diameter, although fines may be treated if sintered. However, it is often more economical to crush coarse ore to less than $\frac{1}{4}$-inch diameter and treat

in a reverberatory furnace. In modern practice, there are few situations in which blast-furnace smelting is clearly the most economical process.

The blast furnace is a shaft furnace that is filled with a vertical column of the charge being smelted. The molten products, that is, slag and matte, are drawn off at the bottom and new material is charged at the top to keep the charge level relatively constant.

Blast furnace fuel is almost invariably coke, which is charged with the ore and flux and ranges in amount from 13 to 17 percent of the charge. The coke burden may be adjusted to control the degree of sulfur elimination, hence it is seldom necessary, unless sintering is required, to roast ores in advance of matte blast-furnace smelting.

Air under low pressure enters the furnace through tuyères near the bottom of the furnace above the hearth. The tuyères are pipes 4 to 6 inches in diameter, spaced 10 to 18 inches apart along both sides and connected with a bustle pipe which takes the air from the blowing engines.

The walls of the furnace usually consist of steel water jackets in the lower protion, and the upper walls may be constructed of refractory brick or may consist of a second tier of water jackets. In copper blast furnaces, the crucible is used as a collecting trough from which the matte and slag flow to a forehearth or settler, in which the slag rises to the top and is removed almost continuously.

<u>Converting</u>

Converting is the final stage in the process of smelting copper sulfide ores or concentrates and is accomplished by blowing thin streams of air through the molted matte (the product of the reverberatory or blast furnace) in a refractory-lined converter.

The first or white-metal stage of converting is rapid oxidation of the iron sulfide in the matte to iron oxide and sulfur dioxide. Enough silica is supplied to form an iron silicate slag. The copper remains as copper sulfide until most of the iron has been oxidized. When slagging is complete, the slag is poured off, leaving molted copper sulfide - known as "white metal" at this stage. Blowing is then continued in the second or blister stage to oxidize the sulfur of the white metal, leaving metallic copper. The first stage is strongly exothermic and raises the temperature

of the bath high enough to form slag and provide enough superheating for the second stage, which does not liberate as much heat.

A low-grade matte produces a larger amount of slag to be re-treated for copper recovery and requires more silica than a high-grade matte; but if the matte is too high in copper and too low in iron, too little heat may be liberated to maintain the process. Heat can be supplied with pulverized coal or fuel oil, allowing the converting of copper mattes, which contain as much as 60 percent copper. All factors considered, from the ore through concentration, smelting, and converting the optimum copper content of mattes in current American practice approximates 35 percent, as in the analysis cited from Morenci practice on page 38.

The two principal types of modern converters are shown in figures 15 and 16. Both types are mounted on trunnions for tilting to receive the charge and pour off slag and metal and both have a row of tuyères in the rear connected with a wind box to introduce air into the charge.

The tuyères are usually 1 to $1\frac{1}{2}$ inches in diameter and 8 to 12 inches apart; they are placed high enough above the bottom to clear the level of the metal at the finish of the blow. Frequent punching of the tuyères is necessary, especially during the blister stage. Each tuyère is equipped with a dyblie ball valve to permit punching of the tuyère without dropping the air pressure and allowing the molten bath to run out through the tuyère.

Air is supplied at 10 to 20 pounds per square inch, and the amount required averages 160,000 to 200,000 cubic feet per ton of blister produced from 35-percent matte.

Two types of converter linings are used in modern practice - basic (magnesite) and acid (silica), basic linings being almost universally preferred, chiefly because of lower operating and maintenance costs. Basic linings have a much longer life than acid linings, converting 2,000 to 2,500 tons of copper per ton of lining as against about 10 to 100 tons with the acid type. Other advantages of the basic lining are the use of larger vessels, the ability to convert low-grade mattes with relatively little lining consumption, and less handling of the shells. The disadvantages of the basic lining are the excessive punching of the tuyères necessitated by formation of magnetic iron oxide about their mouths, the greater time required for lining and repairing, and the excessive blowing out of the fine, siliceous ore during the charging of the flux. Mechanical tuyère punching is employed at some plants.

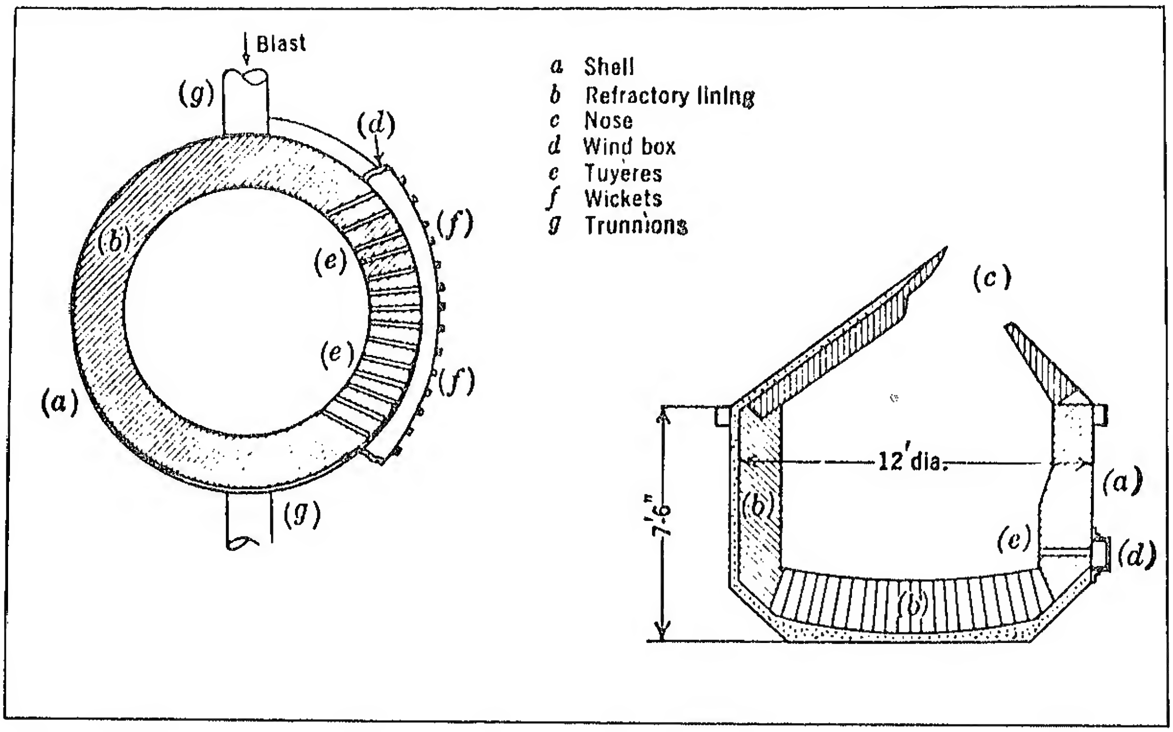


Figure II-15 Upright copper converter (Great Falls Type)

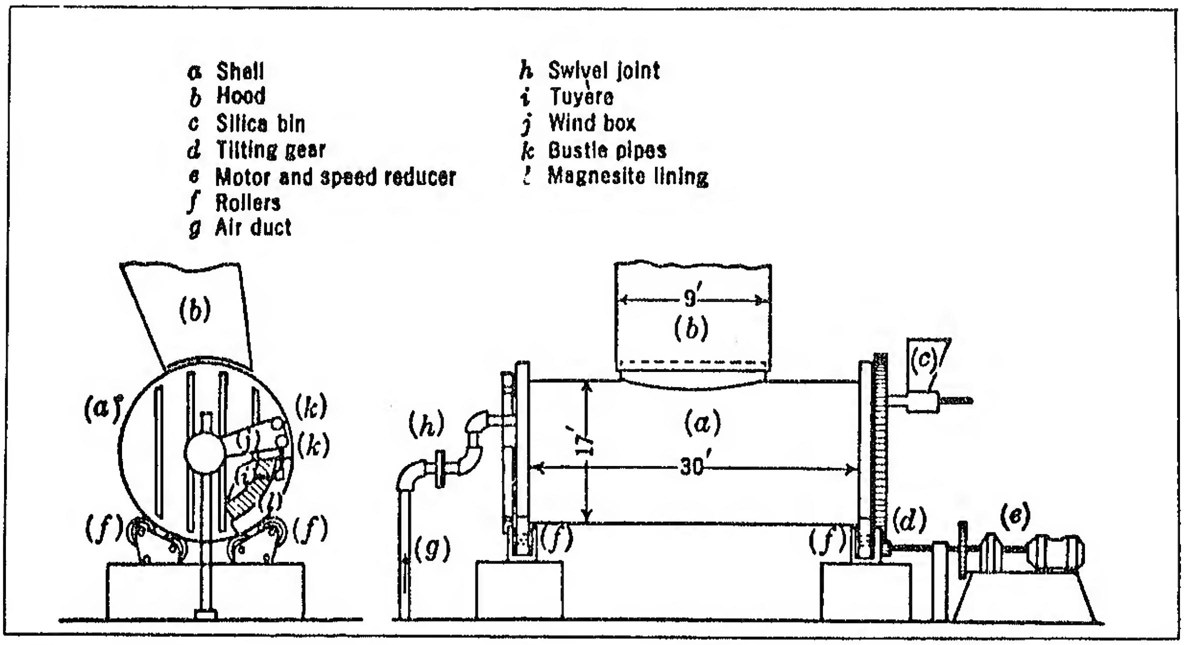


Figure II-16 Horizontal copper converter (Pierce-Smith type)

Converter slag contains 2 to 5 percent copper and is returned to the smelting furnaces for re-treatment, where its high iron content is generally an aid to fluxing.

When the converter cycle is finished the converter is tilted to discharge the copper metal into ladles in which it is transferred to the anode furnace and casting machines. At small plants casting molds are assembled on trucks, which are passed in front of a launder, receiving the copper directly from the converter.

The product of the converter is known as blister copper on account of its rough upper surface when solidified, due to the expulsion of gases, largely air and sulfur dioxide, which saturate the molten metal.

Blister copper contains most of the precious metals of the charge and certain other elements in minor amounts, hence is invariably refined before being marketed.

A 13-by-30-foot Pierce-Smith (horizontal) converter holds 200 tons of matte and at 1.2 cycles for 24 hours has a capacity of about 100 tons of copper per day on 37-percent matte. Within moderate limits, the capacity varies 5 tons with each 1-percent variation in matte tenor. A 12-foot-outside diameter Great Falls (upright) converter may take an initial charge of 10 tons of 40-percent matte and by four successive blows and charges yields 20 tons of blister copper per 6 to 12 hours or 40 to 80 tons a day.

Smelting native copper: Although native copper is found in many deposits, it is only in Northern Michigan and Corocoro, Bolivia, that it occurs in large amounts. Special methods of treatment are generally required. In Michigan the ore is crushed in stamp mills and concentrated. The concentrate was formerly smelted in blast furnaces, but the present method consists almost entirely of treatment in reverberatory furnaces. (See fig. 17.)

The furnaces resemble the cathode refining reverberatories (to be described hereafter) rather than those used for ore smelting. They work intermittently. In the roof are two charging holes, a small one near the fire bridge for fine concentrates and a large one near the center for mass copper. The furnaces are coal-fired, and their capacity may be 100 to 150 tons.

The mineral is charged into the furnace, together with slag from the refining furnace, some coal for reducing, and, if necessary, some limestone flux. It takes about 24 hours to melt the charge and skim the slag, after which the metal is tapped to

FIGURE II-17
COPPER CONCENTRATION

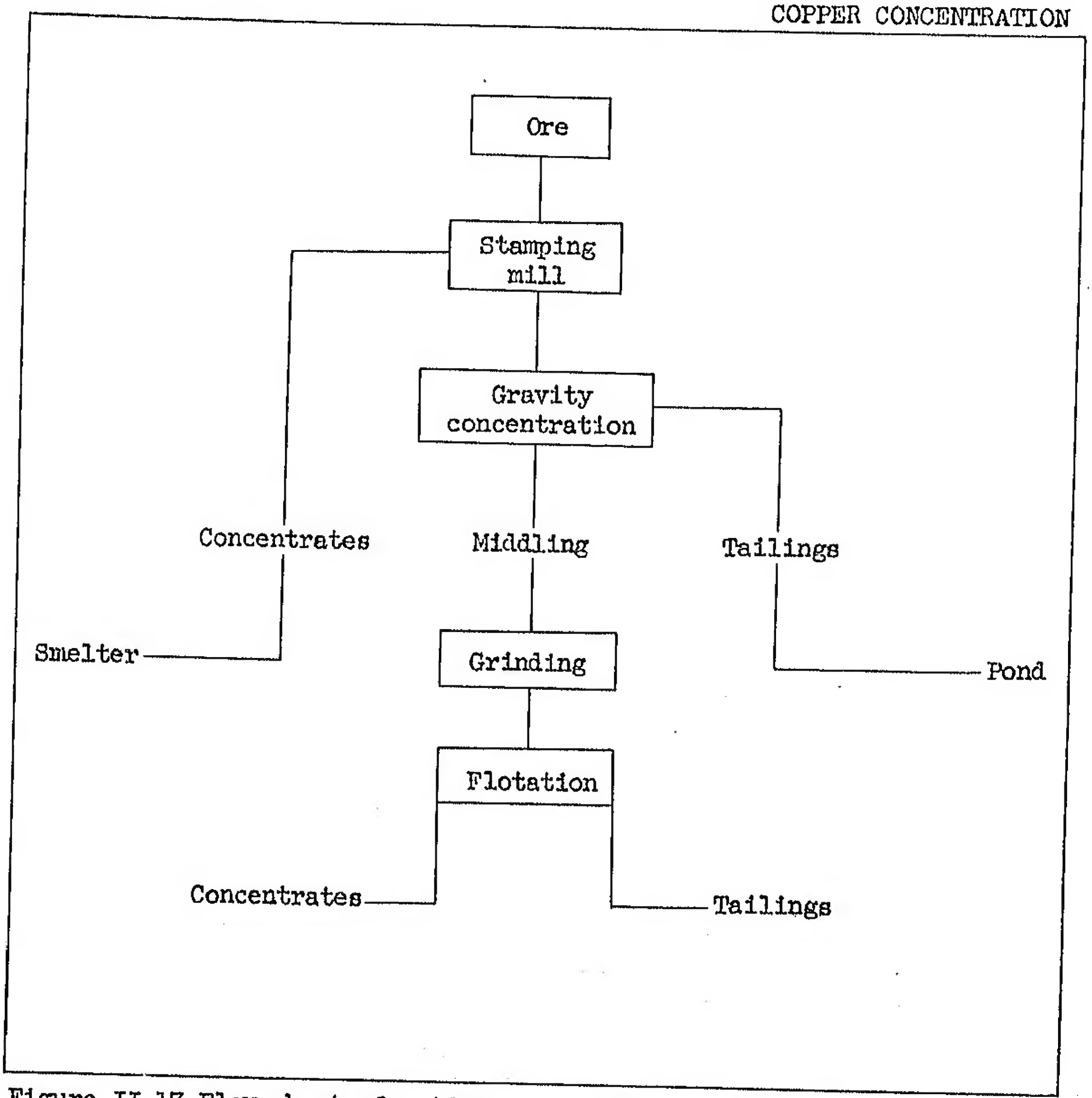


Figure II-17 Flow sheet of native copper concentration

the refining furnace, which is placed at a lower level. The hearth is then repaired with sand and a new charge introduced.

The refining furnaces are similar to the smelting furnaces, except for the charge holes in the roof, which are not required. If the copper is high in arsenic, the charge is treated with soda ash, which is blown below the surface of the copper using about 30 pounds per ton to reduce percentage of arsenic from 0.3 to 0.01. The arsenical slag may be treated to recover the arsenic as calcium arsenate. The oxidizing and reducing operations are similar to those used for treating cathode copper.

Metallurgical Smoke

The metallurgical smoke produced in the roasting, smelting, and converting of copper, as well as in other pyrometallurgical processes, consists of gases, dust, and fume.

Of the gases commonly found in metallurgical smoke - nitrogen, carbon monoxide and dioxide, water vapor, oxygen, sulfur dioxide and trioxide - only the last two are harmful to vegetation. The sulfur compounds often are converted into sulfuric acid, and elemental sulfur is also being produced from flue gases by reduction with coke in a shaft furnace. When sulfur recovery is not feasible, smelter gases after the removal of dust and fume are generally diluted copiously with air and discharged from a high stack at high temperatures to avoid damage to surrounding vegetation.

The amount of dust and its composition depend upon the type of material being treated; but in general dust consists of both original and decomposed or partly decomposed fine particles of ore, flux, furnace lining, and fuel. Its copper content when treating ore may be 7 percent in roaster dust and 25 percent in reverberatory dust to 45 percent in converter dust. A modern reverberatory upon treating, say, 2,100 tons of charge per day may deliver 180 tons of solids in the smoke and roasters; converters also produce considerable quantities of dust, hence the recovery of smelter dust for re-treatment is of importance.

Fume is that part of the solid material in a smoke that has been volatilized or sublimed and subsequently condensed when the gases are cooled in the flue system. The principal constituents are oxides of arsenic, antimony, lead and zinc, sulfuric acid and sulfates. Ordinarily, the fume must be recovered by special apparatus because it is exceedingly fine material; but when recovered, it is mixed with the dust for re-treatment and the mixture is known as flue dust, irrespective of its origin.

The two principal means of separating the solids from the gases before the smoke is dissipated into the atmosphere are baghouses and Cottrell electrostatic precipitators. In some cases one or the other is used, but often both are used together - the baghouse to recover the coarser dust and the Cottrell treater for fine dust and fume. Before the dust-laden gases reach the principal collecting system, it is common practice to pass them through an expansion chamber in which the velocity is somewhat reduced and the larger solid particles drop out and thence into a waste-heat boiler for the utilization of the heat in the gases which often enter the flue at 2,000° F. or higher. From the boilers the smoke is conducted to a baghouse or to Cottrell treaters or in series to both.

A baghouse is a filtering chamber containing a number of cotton or woolen bags made of specially woven cloth (fig. 18). The bags are about 18 inches in diameter and about 30 feet long and are suspended vertically by means of a thimble at the top of each bag. The lower open ends are connected to the gas intake, and the dust-laden gases enter the bags at the bottom and escape through the meshes of the cloth. Dust and fume are caught and held inside the bag, and the cleaned gases pass through. Periodically the bags are shaken automatically from the top. The collected dust drops into hoppers from which it is removed.

The Cottrell process for removing suspended particles from smokes utilizes the fact that, if an electrostatic charge is placed on these particles, they will be attracted to an electrode carrying the opposite charge. Commercial Cottrell treaters (fig. 19) are large chambers containing positive and negative electrodes; the positive electrodes, where the dust collects, have a large surface area and a small radius of curvature as compared with the negative electrode. The positive electrodes are usually pipes or plates, and the negative electrodes are wires or chains and carry a potential difference of 25,000 to 65,000 volts. The accumulated deposits that adhere to the pipes or plates are dislodged by rapping the electrodes at intervals with automatic hammers; the deposits fall into hoppers at the bottom of the treater chamber, from which they are removed periodically.

Virtually any type of suspended material can be removed from a gas stream by Cottrell treaters, and the method has wide applications. It will remove all dust and fume found in copper-smelter smokes, as well as free sulfur trioxide. There is no important copper smelter in the United States or Canada that does not employ Cottrell treaters.

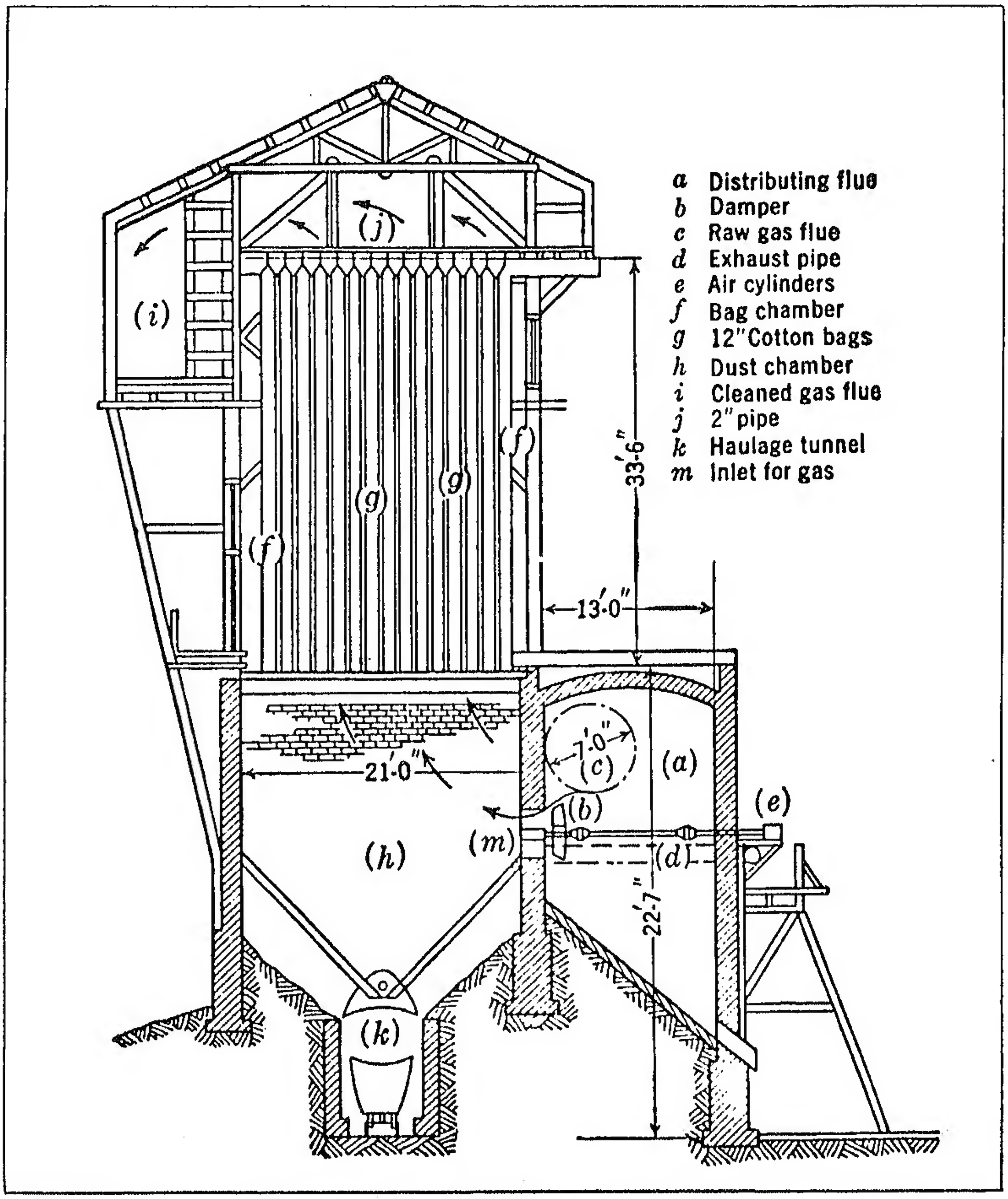


Figure II-18 Baghouse

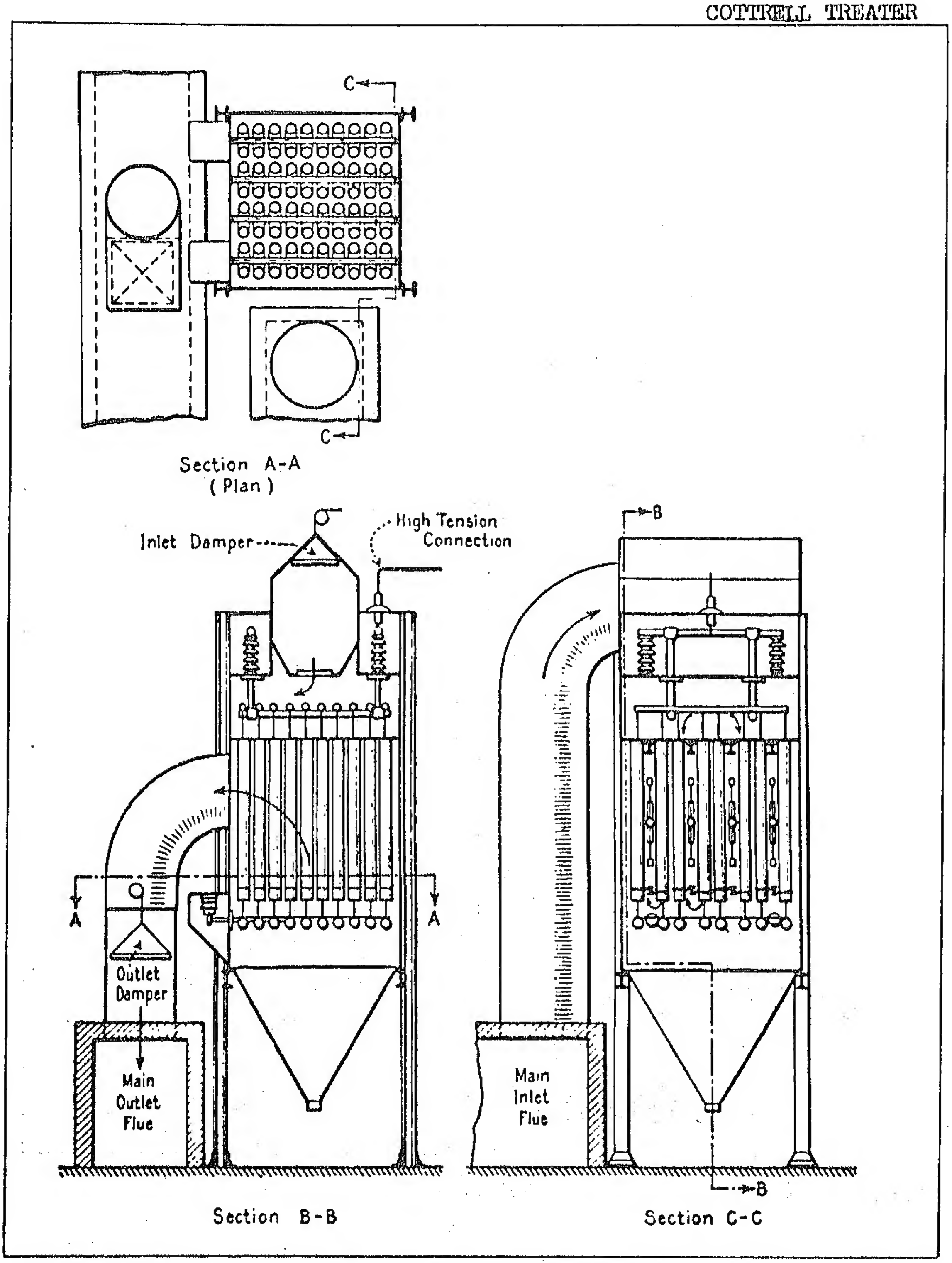


Figure II-19 Pipe type Cottrell treater

The treatment given to the dust and fume collected in copper smelters depends on its composition, but most will be damped, sintered, or briquetted and fed back into the smelting circuit, normally at the reverberatory. The principal byproduct from smelter fume is arsenious oxide or "white arsenic" (As_2O_3), and virtually all the world supply of arsenic is a byproduct of copper and lead smelting. Where arsenic is present in any quantity in the smelter feed, it tends to accumulate in the flue system, because the lower oxide, As_2O_3, is relatively volatile and is driven off in both the roasters and reverberatories. Crude arsenic-bearing dusts are subjected to repeated distillations and condensations until a commercially pure white arsenic is produced, and the residue is then sent back to the reverberatory furnace.

In addition to arsenic trioxide, small amounts of lead and bismuth may be separated in the arsenic plant, unless these dusts contain enough arsenic to warrant special treatment; however, they are usually returned to the smelting circuit, and the contained impurities are removed either in the slag or in the crude copper. (See fig. 20.)

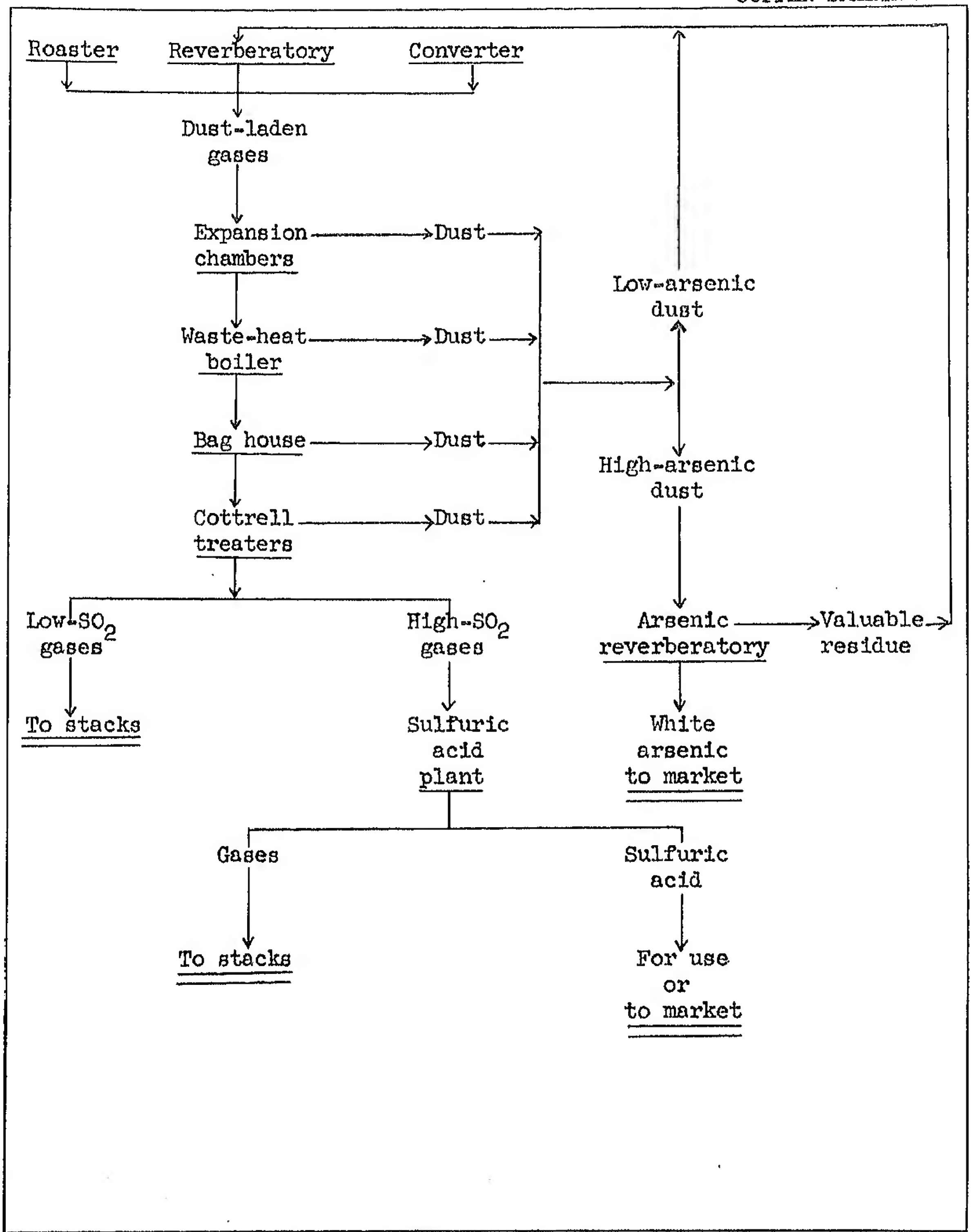


Figure II-20 Flow sheet of copper smelter flue dust and gases

3. Hydrometallurgy of Copper

Hydrometallurgical treatment or leaching is any process whereby the valuable metal or metals are recovered by means of some solvent aqueous, leaving the gangue or waste material virtually unaffected. Leaching is a fairly inexpensive method of recovering copper from low-grade ores; for this reason, it has grown in importance until fully 15 percent of the copper produced in the world is from this method. Fundamentally it is done in three stages - bringing the solvent into contact with the ore or leaching; separating the pregnant liquor from the tailings; and precipitating the metal from the solution.

There are almost as many ways of accomplishing this as there are plants, so a typical flow sheet would be difficult to produce. However, there are certain general requirements that all these methods must meet, such as in the choice of solvents. A solvent must be cheap, attack only the minerals desired, be effective in cold, dilute, preferably water solutions, and should be regenerative so that little new solvent need be added to the cycle.

The kind of ore also determines the solvent used. Native copper ores have been treated economically by ammonia solutions and oxidized and sulfide ores are best treated with sulfuric acid and acidified ferric sulfate, respectively.

Depending upon the disposition of the ore, it may be treated in place, in the mine, in large heaps on the ground, or in specially constructed vats or tubs. Dense ores are sometimes crushed to allow the leaching liquid to percolate through it, and fine slimes are agitated with large mechanical rakes in tanks.

Upon completion of a soak period of 8 days to 3 or more years, the solution is drawn off the ore. It may be drained by gravity, forced out under steam pressure, or sucked out by vacuum. In most cases the ore is washed with fresh water as many as five times, because up to 50 percent of the valuable liquor may remain in the ore on first draining. The original solution and following washes are joined and normally treated to remove iron, chloride and nitrate salts, and molybdenum, which are harmful to the precipitation process. Copper may be precipitated by electrolytic or chemical means. Boiling of ammoniacal solutions precipitates the metal as copper oxide, which is washed, dried, and shipped to a smelter for use in smelting or refining furnaces. The other method, cementation, is displacement of a metal, copper, by a less noble metal, such as iron.

Often it is possible to divert acidic copper mine drainage waters over scrap and recover considerable copper that might otherwise be lost. Scrap iron, usually light scrap such as sheet metal and old tin cans that has been shredded, is placed in tanks and the solution allowed to flow over it. Metallic copper is deposited in flakes on the iron. When enough metal is accumulated, it is flushed or shaken off the iron and sent to a copper smelter. It must be so treated for the metal may contain 10 to 30 percent impurities, including copper oxide. The majority of coarse-size cement copper can be refined by fire refining to ingot copper and used for producing brass and bronze castings. The finer-size portion of the cement is usually charged to the converter. This process has found wide usage, for it is cheap and relatively simple and will work with any strength sulfate solution. The principle drawbacks are the facts that the metal must be refined and that the iron destroys the future usefulness of any acid present. It is used on heap and underground leaching solutions and to strip the last bit of copper from electrolytic deposition plant solutions.

Electrolytic precipitation is the most widely used method of recovering the copper dissolved in an acid solution. The pregnant solution is pumped from the purifying plant to a tank house, which contains large vats similar to electrorefining tanks. Instead of the crude copper anodes, however, insoluble anodes, which act simply as conductors, are used, and the product is of the same high quality as the refined copper. The solution circulates, at a rate of 25 to 200 gallons per minute, through cascaded tanks. Periodically a small amount of solution is removed from the circuit and passes over scrap iron in order to keep the percent of impurities down and also, to strip the liquid of the remaining copper before it is thrown away.

Current densities are far lower in "electrowinning" than in refining, commonly 5 to 13 amperes per square foot of cathode surface, but current efficiencies are also lower ranging from 70 to 90 percent depending on the quantity of ferric iron present in the solution. Power consumption is 8 to 10 times that required for electrolytic refining, as the following example of roughly equivalent conditions shows.

	Electrolytic refining	Electrowinning
	Cell voltage, 0.15V	Cell voltage, 1.8V
Amp. - hr./lb. of Cu	411.	425.
Kw. - hr./lb. of Cu	0.062	0.765
Lb. of Cu/amp.-day	0.0585	0.0565
Lb. of Cu/kw.-day	387.	31.4

4. Refining

Blister copper may contain as much as 5 percent or less than 1 percent impurities, but in any event, too much for most applications of copper, therefore it must be refined. The refining of copper in the United States is done largely on the east coast. Cheap power, proximity to consuming areas, and ocean transportation have combined to produce this concentration. Other refineries are located in Michigan and the Western States. (See chapter VI for locations and capacities of refineries.)

Refining is normally a three-step process of fire refining, electrolytic refining, and finally a second fire refining from which copper may be cast into wire bars of other commercial shapes. The principal exception to this practice is Lake copper, which ordinarily is not electrolyzed.

Fire refining is done to slag off impurities and to oxidize any remaining impurities, which are relatively harmless in that form. The object of electrolytic refining is to remove the precious metals if present and other elements, such as nickel, cobalt, selenium, tellurium, arsenic, and lead, which will not oxidize while copper metal is present. The final fire refining of electrolytic cathodes has for its purpose elimination of sulfur and any other impurities taken up by the copper in the melting operation and in addition, to control the oxygen content in preparation for casting.

Fire Refining

The furnaces used in fire refining are of the reverberatory type, variously known as refining, anode, cathode or wirebar furnaces, depending either on the nature of the charge or the nature of the material being cast. They range from 11 to 14 feet in width and from 26 to 43 feet in length and hold 120 to 350 tons of molten metal. They are constructed principally of siliceous refractories, though magnesite and other basic materials are also used. Because of the danger of breakouts of the heavy molten copper, the bottoms of refining furnaces are built with great care and strength. Refining furnaces may be fired with pulverized coal, fuel oil, or gas. Consumption of fuel per ton of copper produced may vary as follows: 250-300 pounds of coal; 18-21 gallons of oil; 3,200-3,500 cubic feet of natural gas.

The fire refining of native copper is very similar to the fire refining of cathode, blister, or secondary copper: An oxidizing fusion is used to volatilize zinc and lead, to scorify manganese, iron, lead, zinc, nickel, cobalt, and some copper and

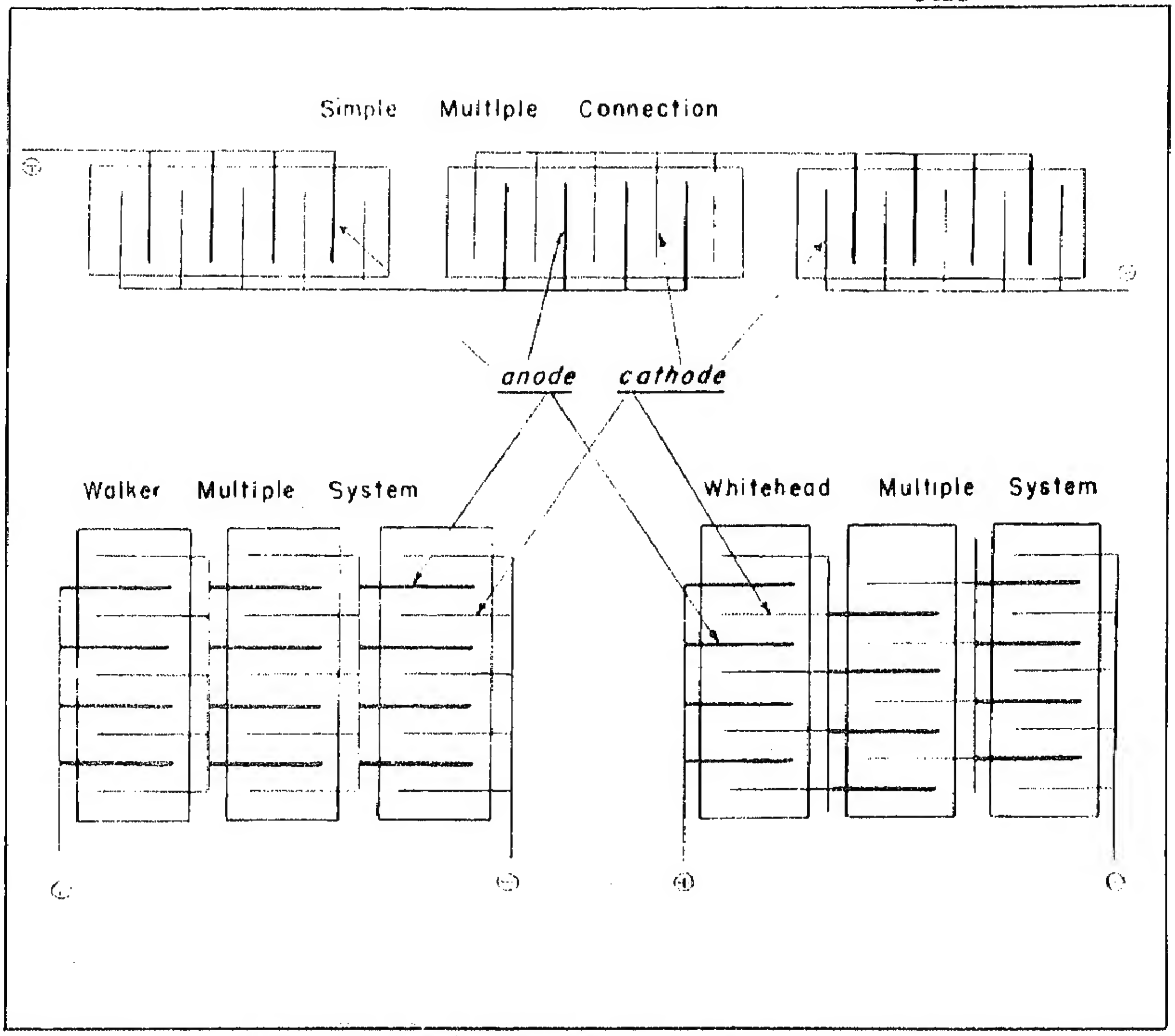
the bath is blown with air to saturate it with cuprous oxide: At this time, the bath contains approximately 6 percent Cu_2O. The dissolved Cu_2O gives up oxygen to the impurities in the molten bath, and these rise to the surface as slag. This slag is skimmed off as rapidly as it is formed. The oxidation must be stopped as soon as the slag formation ceases, because the Cu_2O will then float on the surface like oil, and considerable copper will be lost. This is accomplished by introducing enough reducing gases beneath the bath surface to remove the proper amount of oxygen, which will give the poured copper the "pitch" or "set" desired. Green-wood poles are generally immersed in the bath to form the reducing gases. Following poling, the "tough pitch" copper is cast into anodes for electrolytic refining or commercial shapes for marketing.

Electrolytic Refining

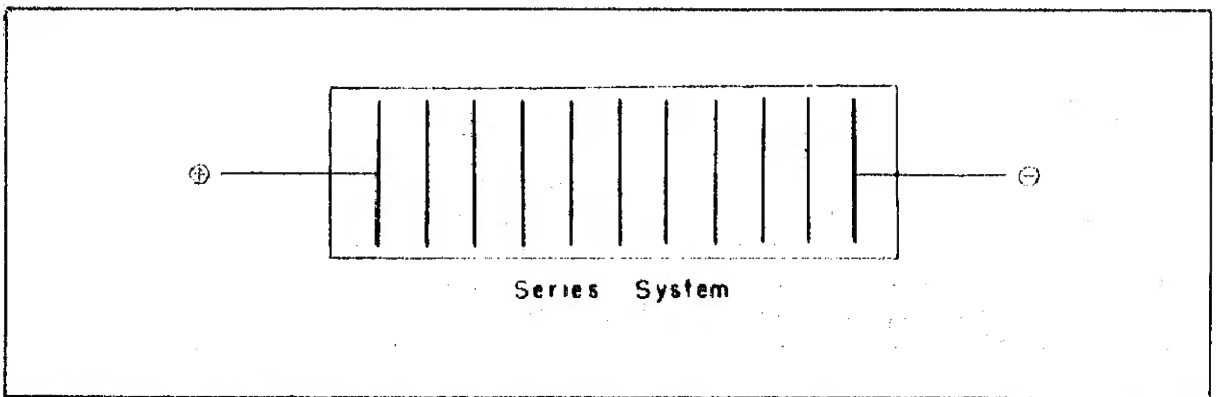
Fire refining, in some cases, will produce copper of high enough purity for commercial use, but in 1950, over 85 percent of the refined copper produced in the United States was electrolytically refined. The widespread introduction of this method was delayed because of the high cost of power and the problems of practical application involved, though the theory of electrolysis is simple. It is the selective transfer of copper atoms from one electrode to another under the impetus of an electrical potential.

The electrical connections in copper refining may be in multiple or in series. Figure 21 shows some of the multiple systems, in which all anodes and cathodes in a given cell have a multiple connection and the cells are connected in series. In all multiple (or paralleled), systems, separate anodes and cathodes are used, and the cathode deposit is built up on a starting sheet of refined copper. The series system uses no starting sheets and the electrodes of impure copper serve as both anode and cathode, the copper dissolving from one side of the anode and the purified copper depositing of the opposite side of the adjacent electrode. The multiple system is more widely used than the series system, largely because of greater flexibility as to the purity of the anode copper and better recovery of precious metals, although the power consumption is greater and more scrap is produced for recycling.

For multiple refining, fire-refined blister copper is cast into anodes about 3 feet long and 3 inches thick, weighing 400 to 700 pounds. Anode molds are so shaped that the slabs have shoulders or lugs by which they can be suspended in the refining tanks. Cathodes, or starting sheets are specially prepared



Multiple system for electrolysis of copper



Series system for electrolysis of copper

Figure II-21 Multiple and series system for electrolysis of copper

electrolytic copper, having about the same area as anodes but being only 1/16 inch thick.

The tanks for refining are built of wood or concrete lined with hard lead and are about 11 feet long, $3\frac{1}{2}$ feet wide, and $3\frac{1}{2}$ feet deep. The electrolyte, a water solution, contains 3 to 4 percent copper and between 10 and 16 percent free sulfuric acid. Tanks are arranged in cascade, so that the electrolyte can be circulated from one to another at a rate of 3 to 6 gallons per minute. Twenty-five to 31 days are required to consume an anode, during which time two or three cathodes per anode will be removed from the tanks.

In the process of deposition, impurities in the anodes may dissolve in the electrolyte, float at the surface as slimes, or fall to the bottom of the tank to make up what is known as anode mud. Nickel and arsenic build up in the electrolyte, causing contamination of the cathodes and increased electrical resistance. To counteract this, some electrolyte is removed from time to time and replaced by fresh solution. The "old" electrolyte is treated to recover the contained metals, and the purified acid is returned to the tank circuit. Some of the solution is drained off and made into sulfate, which is cheaper and more profitable to purify. Anode mud and slimes must also be removed and treated to recover their valuable elements, among which are selenium, tellurium, sulfur, and copper in the form of metal and compounds, plus any gold and silver that may have been present in the anodes.

When enough copper has been deposited on the cathode starting sheets, they are removed and taken to a fire-refining furnace to be melted. Some refining is done to remove any remaining sulfur, but the primary goal of this step is to control the oxygen which brings the copper up to proper "pitch" for casting into desired shape.

Power is supplied from direct-current generators connected with the tanks by heavy copper conductors, or bus bars, in such a manner as to make the impure anodes positive and the pure starting sheets negative. As an example of power required, the Raritan refinery at Perth Amboy, N. J., uses four 1,250-hp. generators, each of which furnishes 396 tanks with a current of 7,200 amperes at 135 volts.

Commercial shapes: Most refined copper appears on the market as ingots, ingot bars, wirebars, cakes and slabs, or billets. These are castings or refinery products. The copper that is sold in semi-fabricated forms, such as rods, bars, tubes, plates and sheets, was originally made from refinery shapes. Refinery shapes and fabricators' products are described in chapter I, section B.

D. FABRICATION

The term fabrication, as applied to the metal industries, refers to the shaping and finishing of metals and alloys from the bulk forms of refined metal or scrap into standard shapes, sizes, temper, and finishes as are required by manufacturers of finished articles or by the construction industries.

Copper and copper-base-alloy products consist of strip, sheets, plates, rods, shapes, tubes, bus bars, commutators, print rolls, and wire and are fabricated in numerous sizes, shapes, and dimensions.

The group of companies manufacturing these products are generally referred to as the "Copper and Brass Industry." A list of the companies fabricating copper and brass in the United States, the location of their plants, and the products manufactured are given in annex II, page 47.

Generalized composite flow sheets of copper fabrication and of copper-base-alloy fabrication are shown in figures 22 and 23.

The wire fabricators are a separate group of manufacturers.

Many types of machinery and equipment are utilized in manufacturing copper and alloy products. They are generally of standard design. In a few cases, specially designed machinery is employed for specific purposes.

The methods of production are more or less standardized, but many vary between two companies producing the same article and also between two plants within the same company or organization.

The fabricators receive their refined metals from the refineries in carlots in the form of cathodes, ingots, billets, slabs, cakes, and bars. These materials are in transit 7 to 12 days, depending upon the point of shipment and the location of the receiving plant.

The scrap used by fabricators originates mostly from processing the various products, the balance being obtained from secondary manufacturers and scrap dealers.

1. Melting Furnaces and Casting

The melting furnaces used almost exclusively by the fabricators of copper and copper-base-alloy products are electric induction furnaces of the low-frequency type. In plate mills, reverberatory furnaces are generally used.

FIGURE II-22
COPPER FABRICATION

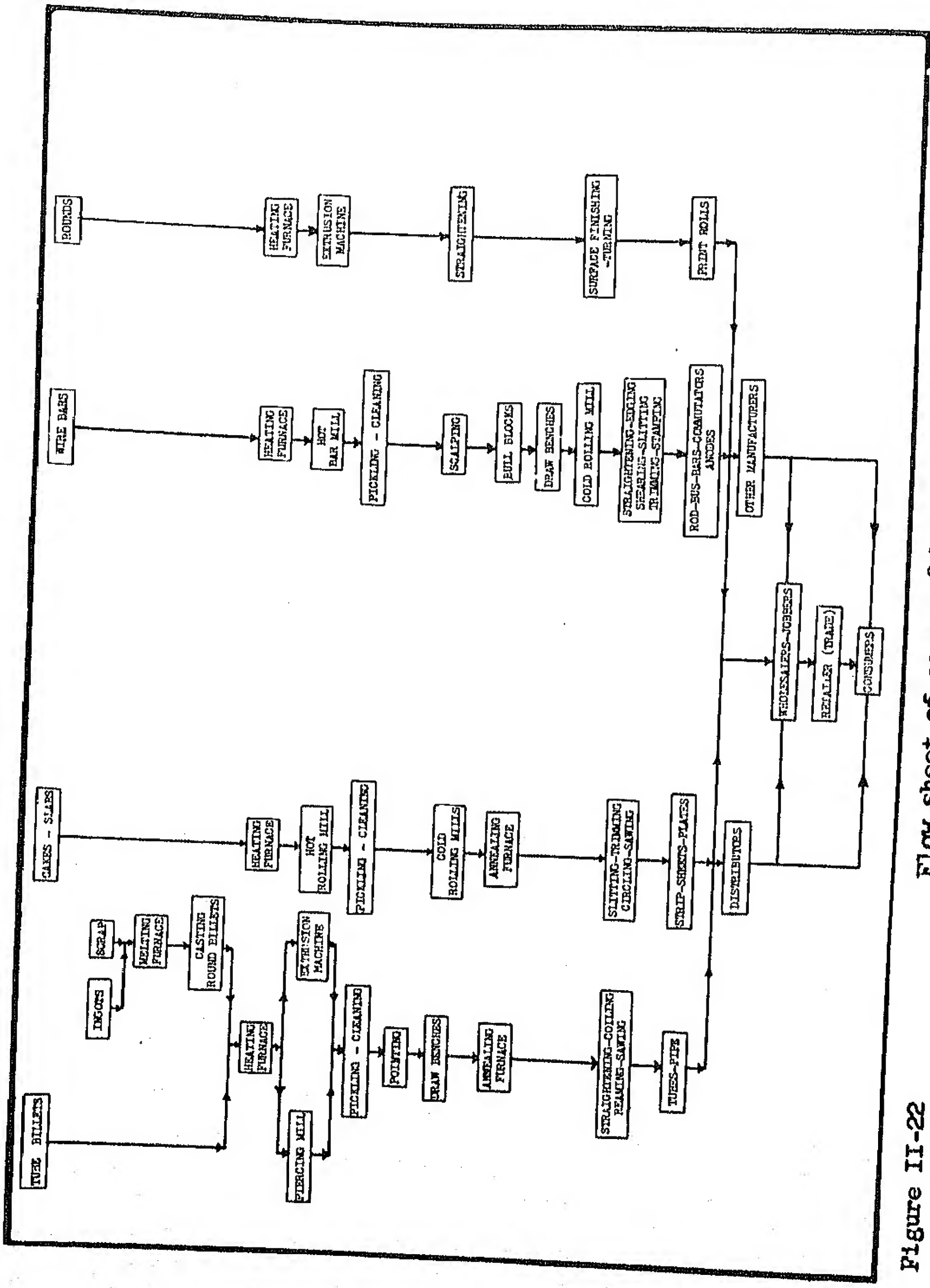


Figure II-22 Flow sheet of copper fabrication

FIGURE II-23
COPPER BASE ALLOY FABRICATION

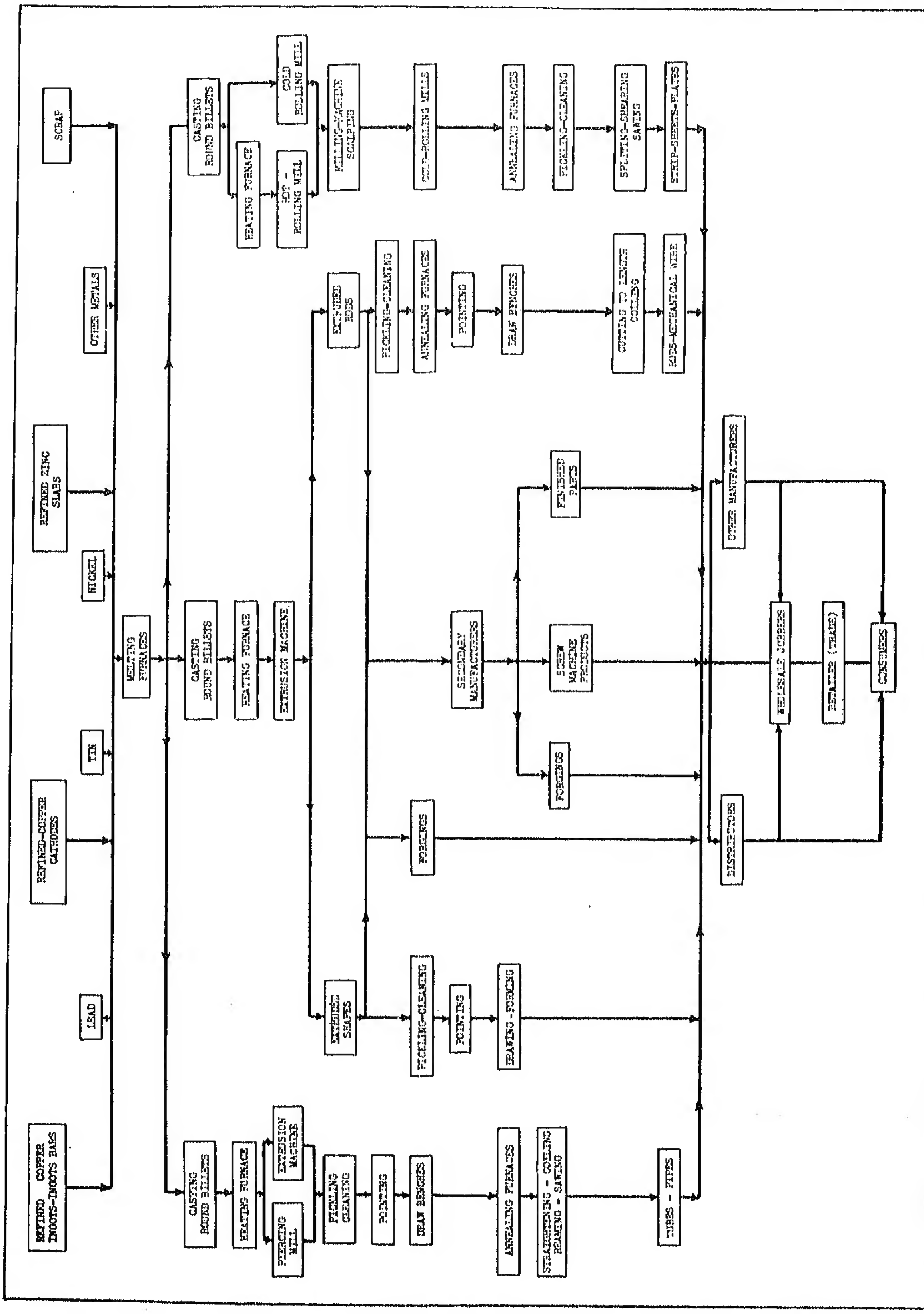


Figure II-23 Flow sheet of copper base alloy fabrication

Induction Furnaces

Low-frequency induction melting furnaces came into use in 1915-20, and soon displaced the older practice of melting in coal- or coke-fired graphite crucibles in all important operations. Induction furnaces are now made in various sizes, ranging from 60 to 350 kw. input and 600 to 5,000 pounds pouring capacity per charge. Those now in general use are rated to about 180 kw. and pour about 1,200 to 2,000 pounds. In some instances double-slot (two inductor tubes serving one bath) furnaces are used to reduce the melting time.

Figure 24 is a diagram of an electric induction, low-frequency melting furnace. Essentially, it is a crucible melting furnace in which a ring of molten metal surrounds one leg of an iron transformer core. The primary winding of the transformer is connected to an alternating-current supply. The current passing through the primary is magnified in the single-turn secondary in direct proportion to the ratio of turns, which in this case equals the number of turns in the primary. This develops an enormous current of about 30,000 amperes, which heats the ring of metal owing to its electrical resistance.

The secondary loop is so designed that the effect of magnetomotive force generated by the high current creates a circulation from the secondary to the crucible and back. Thus the overheated metal in the secondary is continuously replaced by the cooler metal from the crucible and melts the charge with considerable rapidity.

The electrical characteristics of the furnaces depend on several factors: (1) The molten resistivity of the alloy melted, and the resistance of the single turn secondary loop as well as the primary coil; (2) the reaction of both primary and secondary are very important, together with mutual positioning of both with respect to the core.

<u>Refractories</u>: The refractory lining of an induction furnace must have low electrical conductivity to avoid short-circuiting between the primary coil and the molten loop of metal forming the secondary. This precludes the use of silicon carbide, but most other common industrial refractories may be employed, such as fire clay, alumina, magnesia, chromite, silica and mullite. The principal criteria of selection of one or more of these refractories is the refractory cost per pound of metal or alloy melted, which is a function of replacement cost, principally labor, versus operating life.

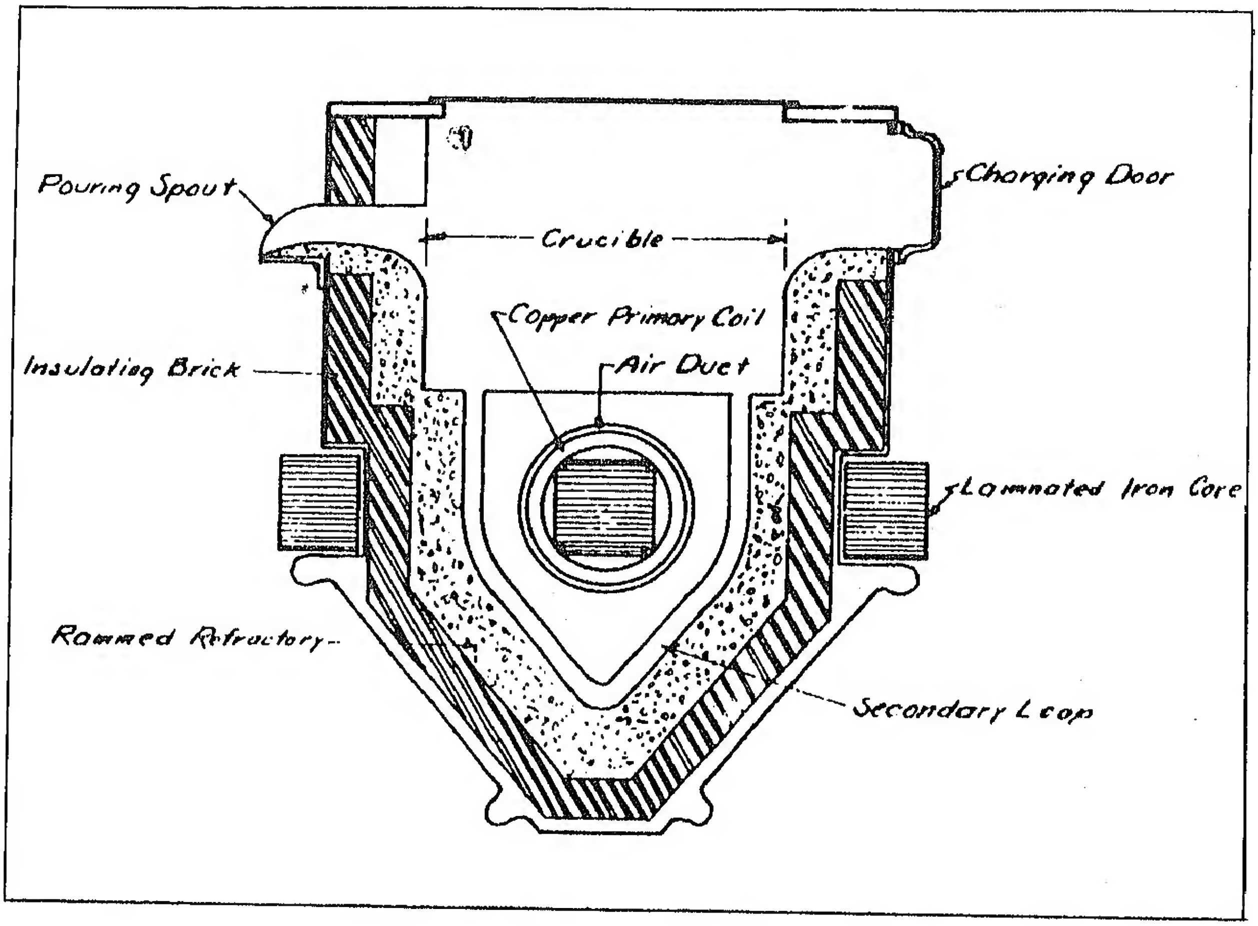


Figure II-24 Induction furnace

A suitable induction-furnace refractory should have good hot strength, low thermal conductivity, high spalling resistance, and good resistance to mechanical and chemical abrasion or erosion.

The life of a refractory in a furnace averaging 600 pounds or more per hour pouring capacity, before it needs to be torn down and replaced, is considered quite good if it pours a half million pounds of some highly refractory alloy, such as cupronickel, though a life 10 times as great is not unusual with some of the more easily melted, low-temperature brasses.

Furnace operation: To get a furnace in operation, linings must be partly vitrified by careful heating with a torch until they are well above a red heat and preferably as high as 2,500° F. This may take 15 hours. About half of a normal charge of molten metal is then added to more than fill the secondary loop and complete the secondary circuit. The current is then turned on, and melting will proceed normally upon addition of unmelted material.

The charging of a furnace is usually accomplished in the following manner: First, a layer of scrap of the same composition as the metal to be poured is placed in the bottom of the crucible. The high-melting constituents of the charge, such as copper, together with part of the zinc and the heavy scrap, are then added so that these will gradually be melted. The balance of zinc is added when the melt has nearly reached its pouring temperatures. When all these are melted, the furnace is skimmed of impurities, dross, etc., through the door and the surface of the metal covered with a shovelful of fresh charcoal to prevent further oxidation. Since the customary pouring temperatures are 200° F. higher than the melting temperatures, the current is left on until a pyrometer in the molten metal indicates that the correct pouring temperature is reached; the charge is then poured.

Due to its chemical activity, zinc acts somewhat as a deoxidizer. When other alloys that do not contain zinc are melted, some other deoxidizer is usually added, such as manganese-copper, phosphor-copper, etc.

All of the molten metal cannot be poured from a furnace of this type, as the secondary ring would be broken; thus, it is customary to pour about 60 percent of the contents. The balance acts as a reservoir of molten metal (commonly referred to as a button), which not only assists in continuing the melting but tends to even up the composition of the heats poured as far as impurities and principal constituents are concerned. The pouring temperatures range from 1,850° to 2,450° F., depending upon the alloy being cast.

In addition to charcoal, fluxes are frequently used to further protect the surface of the metal during pouring. These vary considerably in composition; ordinary table salt is probably the material most generally used, although borax and soda ash are extensively used. A handful of one of these is added to the molten charge, either before or after zinc is added or immediately before pouring.

Reverberatory Furnace

Reverberatory melting furnaces are generally used for casting large slabs or cakes to be fabricated into plates. Although electric furnaces of sufficient capacity are now available, two points detract from their usefulness - the nature of the plant scrap, which could not be used without considerable preparation, such as cutting into small pieces to fit the charging door of such a furnace; and the fact that, in such a furnace, a considerable poundage of molten metal must remain in the furnace to cover the electrodes which makes it difficult to adjust the composition when a change in alloying is necessary.

The hearth of a reverberatory furnace slopes from the charging door downward to the discharge, or tap hole. Because of its location, the complete melt is used, and a change in mixture can be made conveniently.

After discharge, the tap hole is plugged with a mixture of moist fire clay and fine coal. When the melt is ready, a ladle is lowered into a pit under the tap hole and a pointed iron bar is driven through the clay plug. The metal flows into the ladle, and from this is poured into molds of various sizes and styles.

Reverberatory furnaces can be fired with either coal, oil, or gas depending upon the cost and availability.

Casting

Copper and copper-base-alloys are cast in the following forms: Round billets for processing into tubes, rods, and shapes, slabs for rolling into strip, and cakes for rolling into sheets and plates.

Cannon molds: The simplest kind of mold for casting round billets either for use in piercing or extrusion processes is the old cast-iron "cannon mold". This name no doubt originated from the fact that they look like ordinary cannon, which are hung by trunnions from racks and point skyward. The breach consists of

a removable cast-iron plug, and the metal is simply poured into the muzzle. One of the disadvantages of the cannon mold is its extreme bulkiness in the larger sizes.

Figure 25 illustrates an air-cooled billet mold with its mounting on a floor truck. These are for casting billets 9 inches in diameter and about 70 inches long. In this type, the bottom plugs are permanently attached to the truck. The runner boxes are shown in position. After the pouring operation, the mold is lifted off the billet by overhead crane, after which the same crane picks up and removes the cooled billet.

Cannon-type air-cooled molds are generally a single-cavity design and can be made to cast billets 2 to $10\frac{1}{2}$ inches in diameter in varying lengths up to about 75 or 80 inches.

Book molds: Slabs are cast in hinged-type molds, generally referred to as book molds or Lawton-type molds.

Figure 26 illustrates the bottom-hinge Lawton-type mold, $9\frac{1}{2}$ inches wide by about 33 inches long, arranged for clamping with two sets of hooks as shown, complete with hinged runnerbox. For many years, these molds were made with the cast surfaces in the cavity unmachined. It has been found more practicable recently, however, to machine these exactly to size, to obviate the need for grinding or machining the mold surfaces of the billet.

Book molds can be for casting slabs $2\frac{1}{2}$ to 14 inches wide, $1\frac{1}{2}$ to 2 inches thick, and up to about 75 inches long. They are not recommended for slabs weighing over 250 or 300 pounds.

The cast iron used in book molds must be a good-grade gray iron low in sulfur, known as furnace iron. Cast-iron molds tend to absorb moisture and cause "blows" in casting. Furthermore, the growth or permanent swelling of cast iron after repeated use causes cracking of the inner surface of the molds, which results in rough surface castings and internal stresses and contributes to short mold life.

It is the general practice to place a runner box or strainer on top of the molds to receive the metal from the melting furnace. These are generally made of mold iron, but when used with some alloys they are refractory-lined or have refractory inserts. One or more holes of various diameters are drilled in the bottom, depending on the size and shape of the casting.

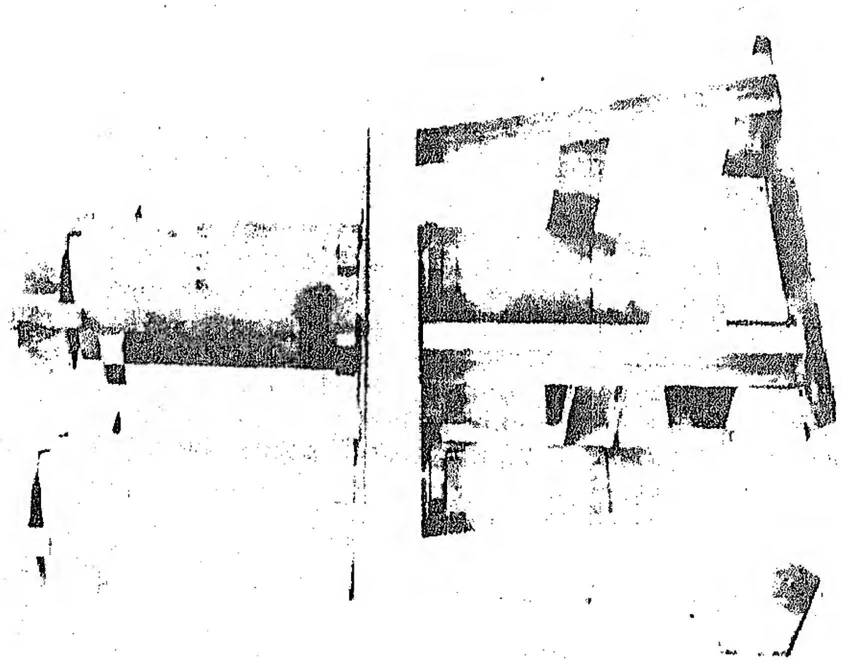


Figure II-25 Air-cooled cannon-type billet mold

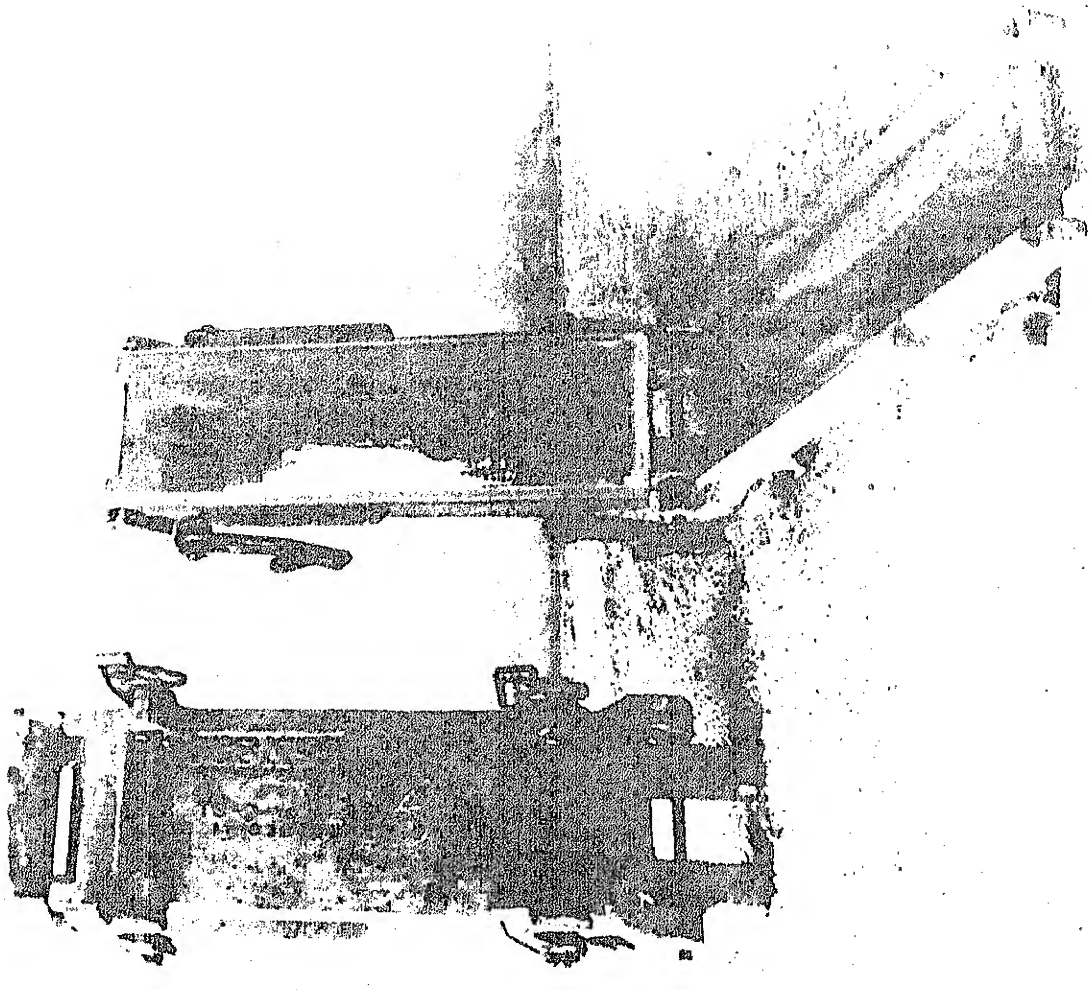


Figure II-26 Lawton-type slab mold

The runner box or strainer controls the rate of flow, directs the pouring stream down the center, distributes the metal evenly across the width of the mold, and keeps dross and dirt from entering the mold. The proper design and use for each particular application are very important.

Production of castings of good surface quality and structure necessitates a suitable mold dressing, which is usually of an oily nature and consists of lard oil, wool grease, tallow or mineral oil, or combinations of these, frequently with additions, such as carbon black, graphite, or silica flour. An important function of a mold dressing is to produce a reducing atmosphere in the mold to prevent formation of excessive quantities of dross.

Too light a dressing or too slow a rate of metal rise in the mold results in a casting with a poor surface. There is a rather fine balance between the rate of pouring and the amount of dressing on the mold.

Heavy cake molds: Molds used for casting heavy cakes to be rolled into plates are essentially of two types - the flat or open-poured and the vertical or top-poured.

The flat-type mold is made of cast iron or copper in various shapes (square, rectangular, or octagonal) and sizes. The design of this type of mold is shown in Figure 27 and is a one-piece casting.

The vertical mold is made of special cast iron in one piece, with four sides but with neither top nor bottom, and is set on a copper stool which acts as a bottom. The sides are made with a slight taper in all directions to assist in removing the casting. Figure 28 shows a vertical mold with hot top built of refractory brick. Hot tops are used when casting metal with a heavy shrinkage and are not generally used on metals with comparatively small shrinkage.

The general rule followed is to use a flat or open-poured mold when casting alloys with a small shrinkage, with or without partitions to alter the shape. Vertical molds are generally used for all metals with heavy shrinkage.

The cast-iron, air-cooled billet and slab molds are being replaced by water-cooled molds, which are simpler to operate and maintain. They also permit casting heavier units.

Water-cooled molds: Water-cooled molds came into use in the early 1930's and in general use in the early 1940's. The weight of a cast slab before the adoption of water-cooled molds averaged

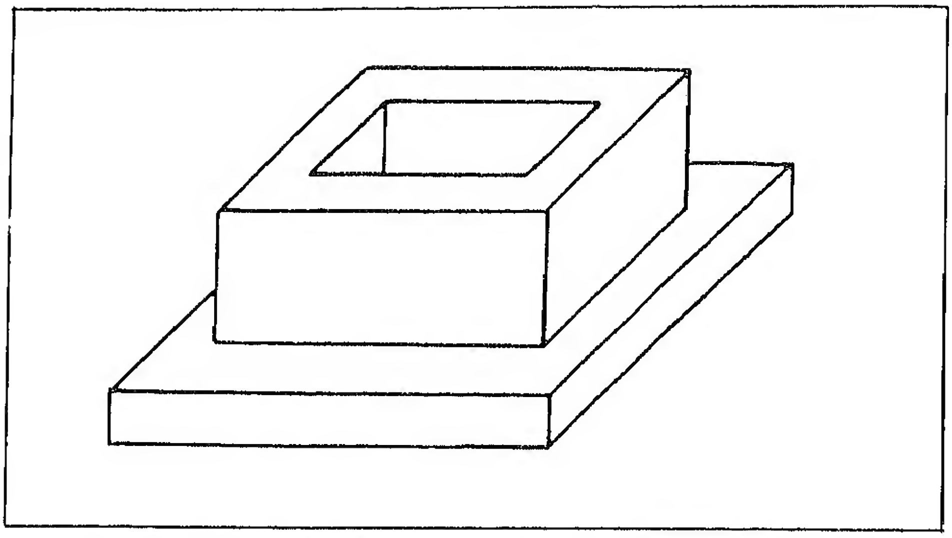


Figure II-27 Flat type or open poured mold

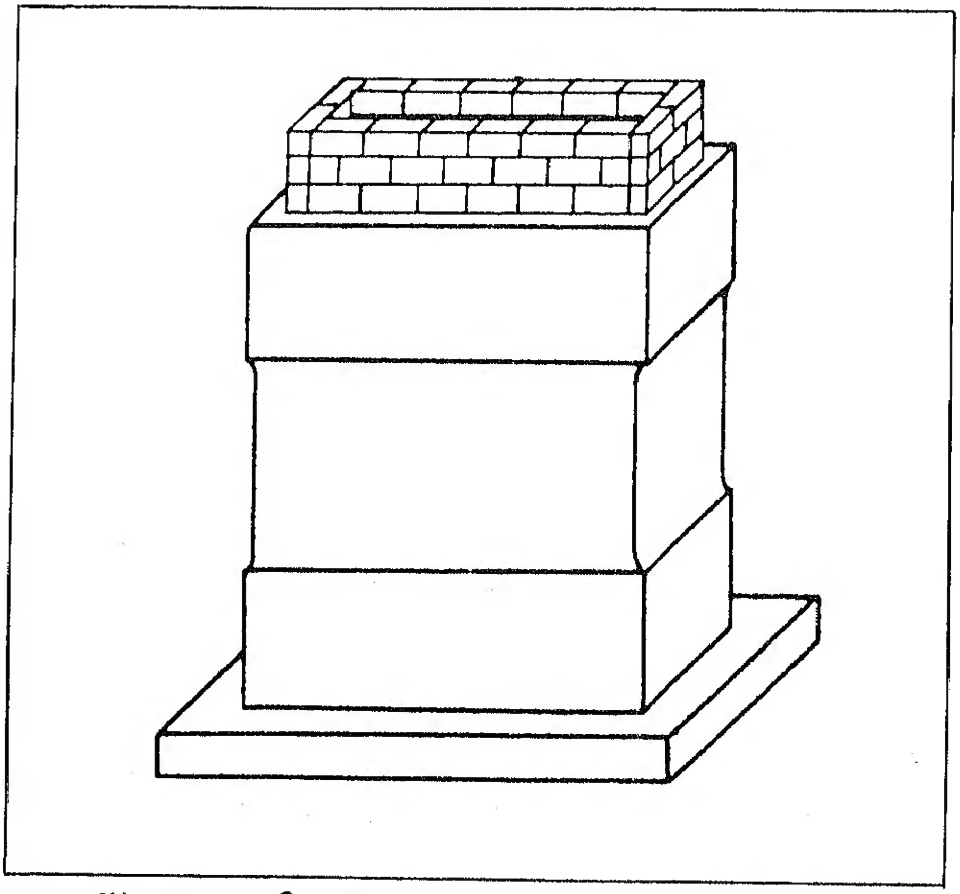


Figure II-28 Vertical or top poured mold

under 300 pounds. The weight of a cast slab for cold rolling now averages 600 to 1,000 pounds and for hot rolling 1,000 to 2,000 pounds.

Water-cooled billet molds are made of cast iron with a cylindrical copper tube insert around which water circulates.

Figure 29 illustrates water-cooled molds for billets about 3 inches in diameter by 75 inches in length arranged with two cavities per mold. These are shown complete with piping, trunnion, and trunnion stand, ready for mounting in a shop.

Water-cooled billet molds are made for sizes from 2 to 10½ inches in diameter and from 50 to 115 inches in length. For 2 and 3 inches diameter they can be made in double- or multi-cavity designs, whereas the larger diameters are generally single-cavity designs.

The molds are mounted vertically during pouring and tipped horizontally for unloading.

Slab molds have copper face plates about 1 inch thick, held in a cast-iron water jacket and open like a book. Figure 30 illustrates water-cooled slab molds. These molds are of the side-hinge type and are shown complete with a runner-box of the set-on type and piping.

The water-cooled mold design of the Junker type, is made to cast 500- to 5,000-pound slabs in widths up to 42 inches, thickness 2¼ to 6 inches, and in lengths up to the longest, 96 inches cavity.

Water is circulated through the molds at the rate of approximately ½ gallon per minute per pound of metal cast. The temperature of the inlet water is controlled and ranges from 100° to 160° F., depending on the size and shape of casting, on the alloy, and on plant practice.

Continuous casting: Briefly stated, continuous casting can be defined as a process wherein liquid metal enters one end of a mold continuously at a substantially uniform rate and solid metal emerges simultaneously from the other end.

Virtually all the tonnage in production is confined to three processes, each of which produces castings of any desired length without stopping. The three processes - stationary mold, vibrating split-mold, and reciprocating-mold - have several factors in common:

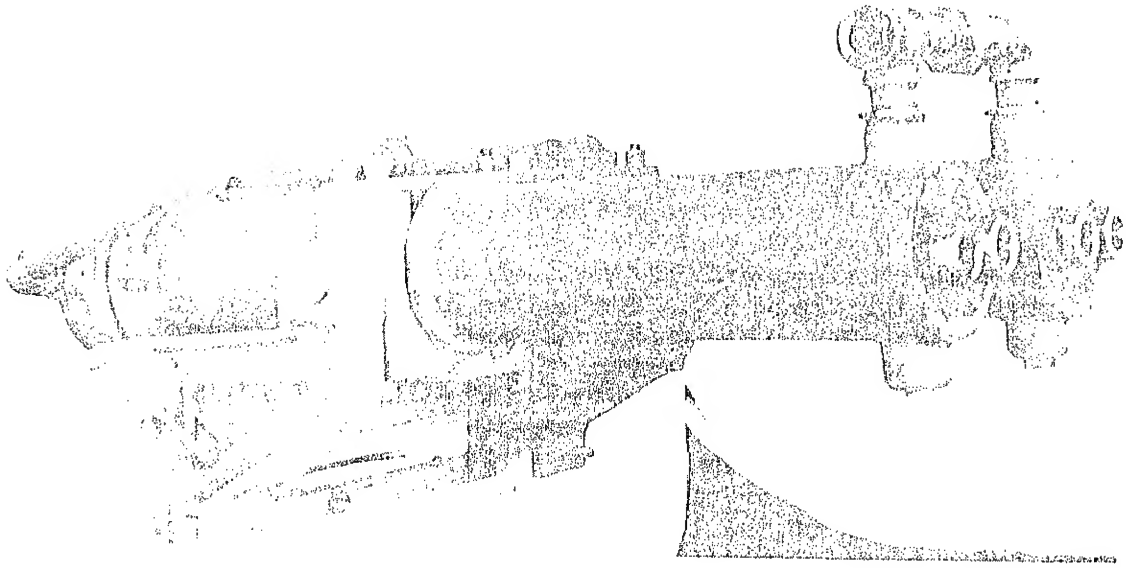


Figure II-29 Water cooled billet mold

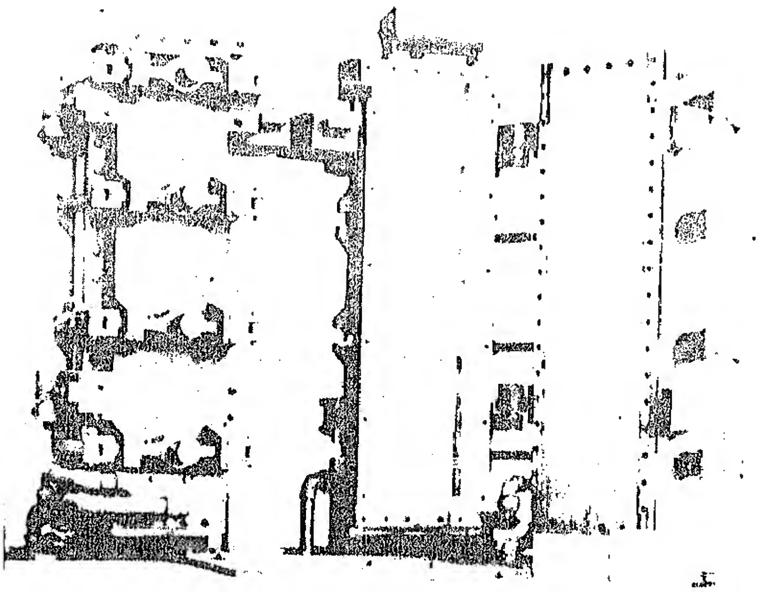


Figure II-30 Water cooled slab mold

Water-cooled copper, copper-alloy, or graphite molds; mechanical means for withdrawing or controlling the billet speed through the mold; and traveling saws to cut the casting into desired lengths without stopping the continuous pouring.

The stationary mold process is one in which the mold remains fixed; the molten metal enters at the top and emerges at the bottom as a solid, continuous casting. A mold of this type is generally in one piece and rather short. This process is used for copper and alloys difficult to cast by more conventional methods.

The vibrating split-mold process consists of two water-cooled mold sections that together form the contour of the casting. These sections are mechanically opened and closed laterally, with a very small gap and at high frequency. The molten metal, entering at the top, solidifies, is withdrawn at constant speed, and is then cut into the desired lengths by integral saws. This process is used principally for casting copper.

In the reciprocating-mold process, the liquid metal is underpoured from a reservoir furnace through a downspout and discharged underneath the surface of the liquid metal in the mold. The mold has a downward and upward motion. The downward motion is synchronized with the speed of the billet, and the upward speed is about three times that of the downward speed. The total travel of the mold is about 3/4 inch in each direction. This process is used principally for copper-zinc alloys.

In the three foregoing processes, further cooling is necessary after solidification. This subsequent cooling is either by direct application of water sprays or by secondary cooling jackets.

The linear casting rate of these machines varies from 6 to 20 inches per minute, depending on the alloy and cross-sectioned area. The principal application of continuous casting is in large units, but the process is also applicable to small units for some alloys difficult to cast by other methods.

The metallurgical advantages of these processes are less segregation, high density, and absence of shrinkage porosity. Operational advantages include these: Straight-line, continuous production, lower-pouring temperatures required, elimination of butt or gate scrap, exact lengths delivered to the mill, greater uniformity of composition, and general over-all economies. Disadvantages are the high initial cost and the lack of flexibility of alloy charges.

The bulk of the tonnage produced by continuous casting is in the form of billets, slabs, and rod for standard fabricating processes.

2. Heating and Annealing

In the fabrication of copper and copper base alloys products the metal may be heated for hot working, for reworking, for strain relief or to obtain the final temper.

No specific rules can be made for establishing hot-working or annealing temperatures. They can be formulated only after all contributing factors of each job are known and considered, as affected by variations in equipment and operating methods.

Heating preparatory to hot-working usually requires higher temperatures that annealing for cold working. Most metals and alloys have a relatively wide hot-working range, but some have a narrow range. The working temperature depends on the material and the type of operation and may vary from about 1,200° F. to 2,000° F. At these temperatures, the metal is more plastic than at atmospheric temperature.

Annealing consists of heating metals between cold-working operations, usually after a 30-percent or greater reduction has been made either by drawing or rolling. The cold working of metals results in strain and distortion of the grain structure with an accompanying increase in hardness and a decrease in ductility. In progressive cold working, a point is reached at which further deformation cannot be made economically or without structural damage to the metal. To restore the ductility for further cold working, the metal must be annealed by heating to a temperature where it recrystallizes and the grain is changed to the proper size. Annealing is also done after the last stage of cold working to obtain the desired temper or hardness or to relieve the strains in the metal to eliminate season cracking. This is generally referred to as "relief annealing".

It is impracticable to anneal for definite properties of tensile strength or hardness between the normal cold-worked and the fully recrystallized or softened range, because of the extremely rapid rate of change of properties with only a small change in metal temperature. Rates of heating and control of temperature are not accurate enough for obtaining any desired tensile strength or accompanying physical property. The best practice is to anneal completely and then to obtain the desired properties by controlled cold working to the necessary extent.

When copper that contains oxygen is to be annealed, the hydrogen in the atmosphere must be kept to a minimum to reduce the danger of embrittlement caused by combination of the hydrogen in the atmosphere with the oxygen in the copper, forming water vapor under pressure, and resulting in minute ruptures in the metal.

Commercial copper is one of the easiest metals to anneal and yet maintain a clean bright surface. Copper of course, will readily oxidize at elevated temperatures and form cuprous or cupric oxide. Cuprous oxide is reddish or rose color and cupric oxide is jet black.

No preparation of the metal to be heated for hot working is necessary, as it is in the cast or rough form, such as billets, slabs, cakes and bars.

Materials are annealed in various forms. Tubes and rods are annealed in straight lengths and in coils, strip metal in flat strips and in rolls, and sheets and plates in flat form. Wire is annealed in coils or on spools and in some instances in strands.

It is usually not practicable to anneal a variety of different sizes or kinds of material in the same charge, because of different rates of heating and variable final temperatures.

It is considered good practice to deliver materials to the furnace to be annealed clean, free from excess oil or lubricants. Regardless of the type of furnace used or the article to be annealed, it is advisable to eliminate as much of the drawing or rolling oils or lubricants as possible before annealing, as they may cause staining that is very difficult to remove.

It is extremely important that oil, or any other lubricant used in rolling or drawing, contains no appreciable amount of fixed alkali as, for example, in lubricating soaps. Such alkali is likely to be set free at higher temperatures and cause considerable etching of the surface. This is more pronounced in the case of the brass than copper, but fixed-alkali soaps should be avoided in annealing any of the nonferrous alloys.

Excessive sulfur in the lubricant or fuel will cause discoloration of the metal; red stains appear on yellow brass and reddish brown on copper rich alloys.

Furnaces

Many types of equipment are utilized for heating and annealing. Selection of the size and type of furnace for a given application

depends primarily on the particular heating process involved, the expected rate of production, and the most efficient means of moving the material to, through, and from the furnace.

The fuel may be oil, gas, or electric power, depending on cost and availability and the design of furnace. Gas or electricity is preferred, where economically available, because closer control of temperature is possible and less surface scale or oxides are formed, resulting in less pickling and cleaning.

Heating furnaces: Furnaces for heating before hot working are selected on the basis of the forms to be heated and are usually side-fired, with oil, gas or combination burners.

Furnaces for heating round billets may be the simple roll-down type, the conveyor type, or the push-through design. The hearth in the roll-down furnace is built on a slope, which allows the billets to roll from the charging end to the delivery end by gravity.

Figure 31 shows a conveyor-type furnace for heating piercer billets 3 inches in diameter and 50 inches in length. It also shows the selective feeding mechanism, in which billets are laid down in bundles on rails adjacent to a chain-operated lifting device, incorporating star wheels for selective disposition of billets one at a time to the screw fed conveyor. This type of furnace is built to handle billets up to 8 inches in diameter. The billets are moved through the furnace by a screw-feed conveyor, and the surface of the billet is continuously exposed to the heat, thereby assuring uniform heating.

In push-through furnaces, the billets are placed on rails or trays and pushed through by means of hydraulic operated pushers. Furnaces for heating slabs or cakes are generally of this type. The hearth construction of these furnaces incorporates heat-resisting alloy rails or bars on which the slabs or cakes rest, and hydraulic operated pushers are employed at the charging end for pushing the material through the furnace. At the discharge end, the material is removed from the heating chamber, manually or mechanically.

Figure 32 shows a push-through-type furnace for heating copper wire bars. The charging equipment shown consists of a chain conveyor on which the bars are placed. As the conveyor moves forward, the leading bar is placed in a selective jaw-clutch feeding device, which in turn delivers the bar to a table in front of the charging end of the furnace. Hydraulic operated pushers are employed at the charging end for pushing the material through the furnace. The bars

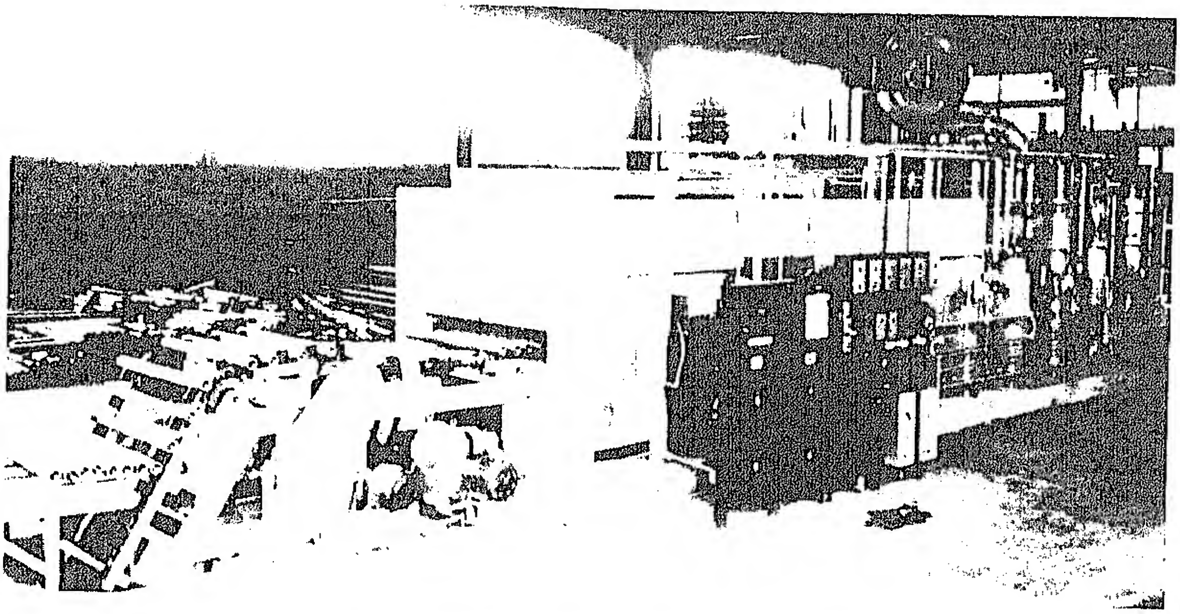


Figure II-31 Conveyor-type billet-heating furnace

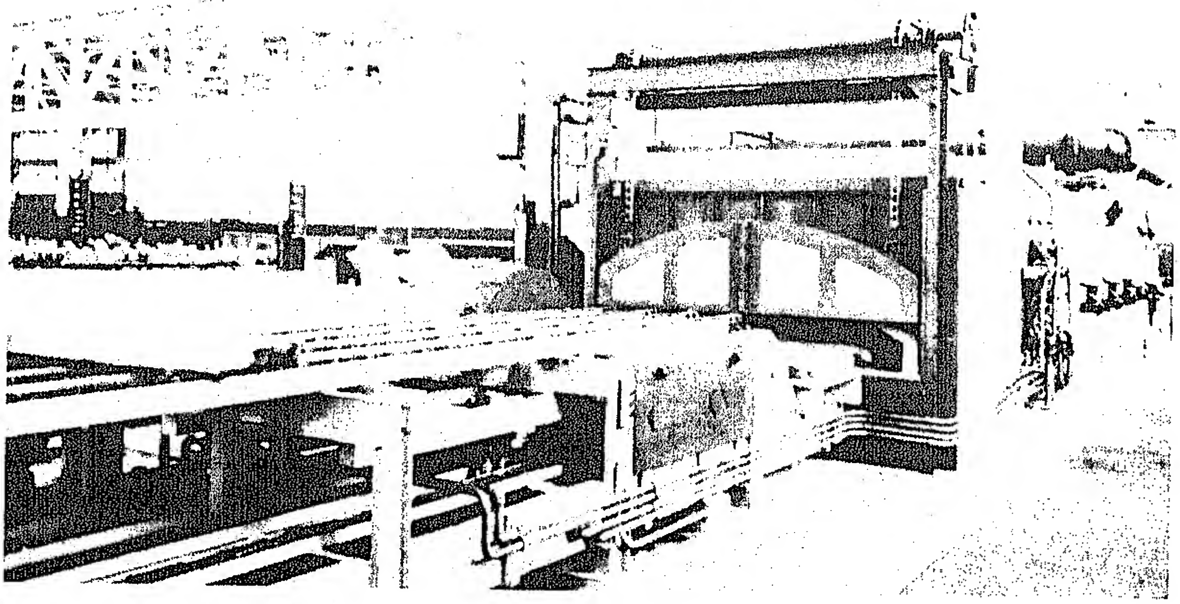


Figure II-32 Push-through bar-heating furnace

rest on heat-resisting alloy rail while being pushed through the furnace. At the discharge end an extractor mechanism is utilized to remove the heated bar from the furnace. The heated bar is conveyed or is carried on a buggy to the rolling mill.

Annealing furnaces: In general, annealing furnaces are either of the batch or the continuous type. The nature of the material being handled, tonnage requirements, and operating conditions ordinarily will determine the type to be employed.

In the batch furnace, the materials usually are loaded on pans in comparatively high and broad stacks and pulled in and out of the furnace by cables. In such stacks, under ordinary conditions, it is obvious that the outer layers are the first to reach temperature, with the center of the stack last. Conversely, on cooling the outer layers are the first to cool. This results in a longer total heating and cooling cycle than is required in a continuous furnace.

In the continuous furnace, the material is generally loaded one layer high and conveyed through the furnace by means of driven rollers or other methods of conveying. This results in uniform distribution of heat above and below the metal.

An increasing number of furnaces is being used with special atmosphere-controlling equipment. When a specially controlled atmosphere is used during annealing of copper and alloys, a much cleaner metal surface is obtainable, resulting in a lower cost for cleaning and pickling. Controlled atmosphere may be produced with many different types of equipment, depending on the desired results. For bright or clean annealing an inert-gas atmosphere is usually employed.

Copper may be bright-annealed in an atmosphere of pure steam or clean-annealed in an atmosphere consisting entirely of products of complete combustion.

Furnaces for clean or bright annealing may be divided into two general classes from the standpoint of atmosphere application. The type frequently used consists of a tightly closed chamber filled with special inert atmosphere generated in an external gas unit. With this type of equipment, absolute control of the composition of the atmosphere within the furnace is possible at all times if proper sealing means are provided for the doors.

A furnace of this type ordinarily employs an entering and leaving vestibule, with interlocking doors at each side or a thermal trap so arranged that the point of entrance is well below the level of the hot gases within the furnace.

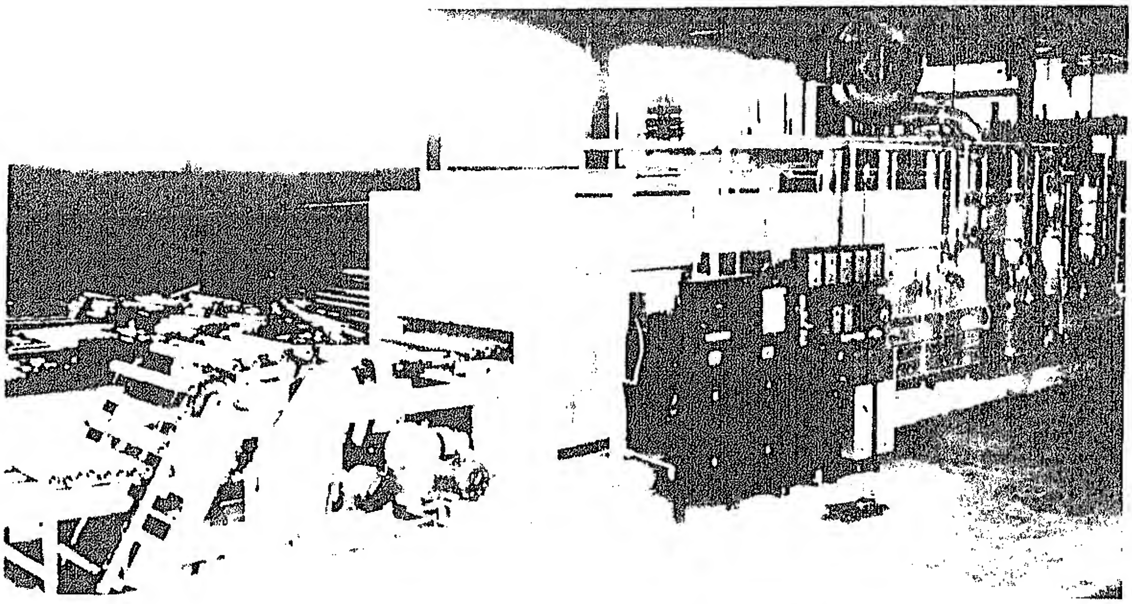


Figure II-31 Conveyor-type billet-heating furnace

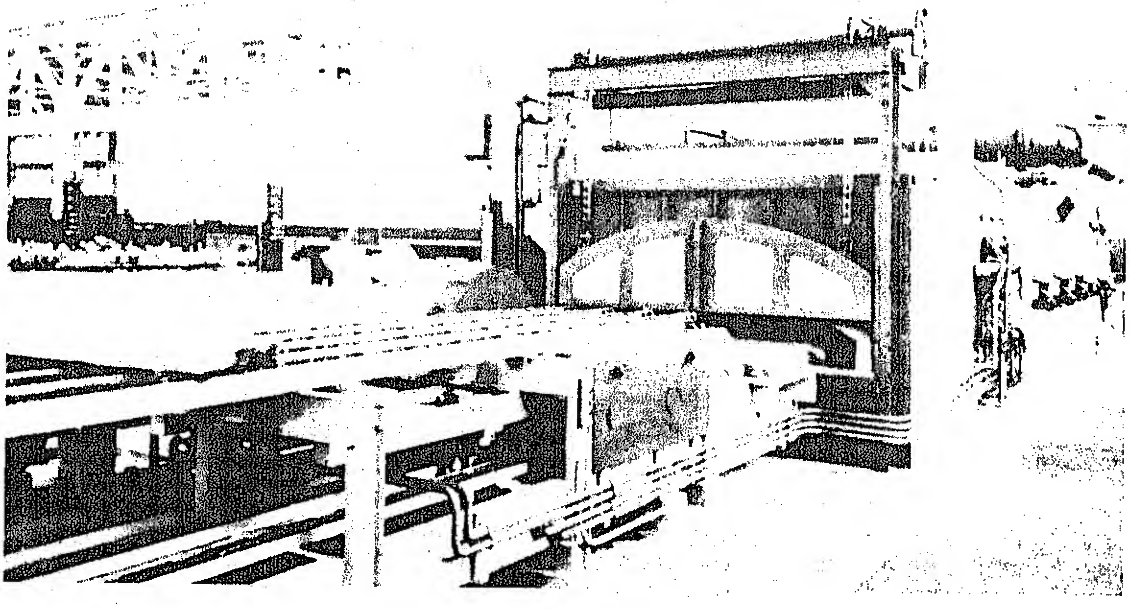


Figure II-32 Push-through bar-heating furnace

rest on heat-resisting alloy rail while being pushed through the furnace. At the discharge end an extractor mechanism is utilized to remove the heated bar from the furnace. The heated bar is conveyed or is carried on a buggy to the rolling mill.

Annealing furnaces: In general, annealing furnaces are either of the batch or the continuous type. The nature of the material being handled, tonnage requirements, and operating conditions ordinarily will determine the type to be employed.

In the batch furnace, the materials usually are loaded on pans in comparatively high and broad stacks and pulled in and out of the furnace by cables. In such stacks, under ordinary conditions, it is obvious that the outer layers are the first to reach temperature, with the center of the stack last. Conversely, on cooling the outer layers are the first to cool. This results in a longer total heating and cooling cycle than is required in a continuous furnace.

In the continuous furnace, the material is generally loaded one layer high and conveyed through the furnace by means of driven rollers or other methods of conveying. This results in uniform distribution of heat above and below the metal.

An increasing number of furnaces is being used with special atmosphere-controlling equipment. When a specially controlled atmosphere is used during annealing of copper and alloys, a much cleaner metal surface is obtainable, resulting in a lower cost for cleaning and pickling. Controlled atmosphere may be produced with many different types of equipment, depending on the desired results. For bright or clean annealing an inert-gas atmosphere is usually employed.

Copper may be bright-annealed in an atmosphere of pure steam or clean-annealed in an atmosphere consisting entirely of products of complete combustion.

Furnaces for clean or bright annealing may be divided into two general classes from the standpoint of atmosphere application. The type frequently used consists of a tightly closed chamber filled with special inert atmosphere generated in an external gas unit. With this type of equipment, absolute control of the composition of the atmosphere within the furnace is possible at all times if proper sealing means are provided for the doors.

A furnace of this type ordinarily employs an entering and leaving vestibule, with interlocking doors at each side or a thermal trap so arranged that the point of entrance is well below the level of the hot gases within the furnace.

Unless careful provision is made, however, vapor from roll or drawing lubricants and other impurities are likely to accumulate within the heating or cooling chambers, and considerable discoloration to the outer surface of the metal may be experienced. Proper venting will do much to overcome this difficulty.

The other type of clean-annealing furnace depends on control of the combustion equipment for generating an inert atmosphere. Furnaces of this type have the advantage that the end of the furnace may be kept open for loading and unloading as the outgoing products of combustion will prevent entrance of an excessive amount of room air to oxidize the metal. Furnaces of this type, in which the outgoing metal is cooled with a water spray, are very economical and quite rapid in operation. The surface of copper articles passed through such a furnace may be controlled quite accurately to give any desired finish.

Figure 33 shows a furnace for bright-annealing of copper or clean-annealing of alloy wire in bundles or on spools. Bulkhead-type trays are used, which eliminate the use of doors at the charging and discharge end of the equipment.

Figure 34 shows a bright or clean annealing furnace for tubes in coils or straight lengths. This furnace is equipped with a dry-cooling chamber. If this type of furnace is used for intermediate annealing only it is common practice to utilize a water-quenching chamber in place of the dry-cooling chamber.

Cooling: To obtain a clean or bright surface on annealed metal, the material must be cooled before being exposed to the air; this is generally done in a separate chamber after leaving the heating chamber.

Cooling is best accomplished by rapidly circulating the furnace atmosphere through cooling devices around the material. Special water-cooled radiators have been developed for this purpose and have proved highly satisfactory. With the proper equipment, cooling may be accomplished in approximately twice the time required for heating. Some of the less-efficient cooling devices require greater intervals, up to two and one-half or three times the heating period.

Recirculating furnaces are being utilized more extensively for annealing nonferrous products, to obtain a clean surface, closer control of temperature, and lower fuel cost. This type of furnace is generally used for low-temperature annealing (usually under 1,200° F.) where gas or oil is burned in a separate chamber and the resulting products of combustion, together with air, are circulated

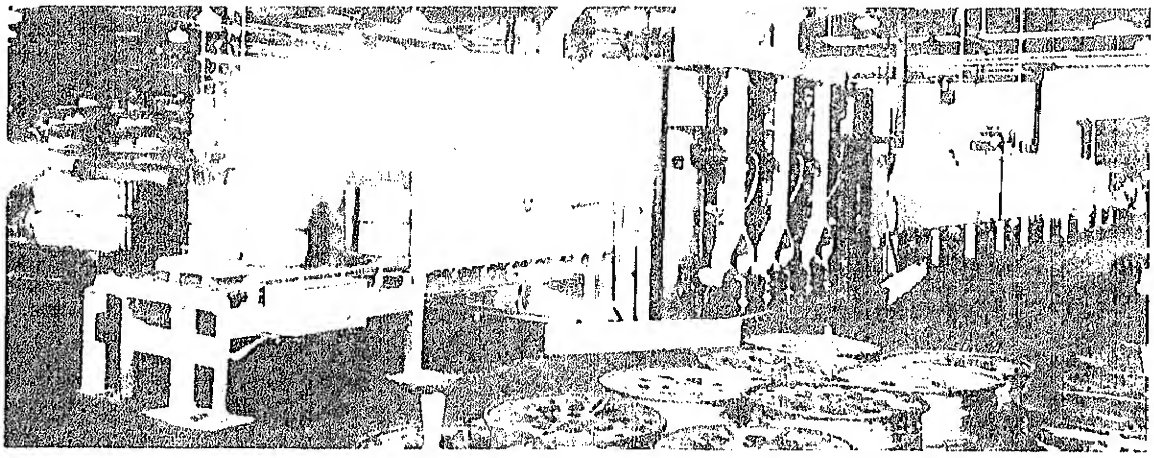


Figure II-33 Single-deck, roller-hearth, tube-annealing furnace

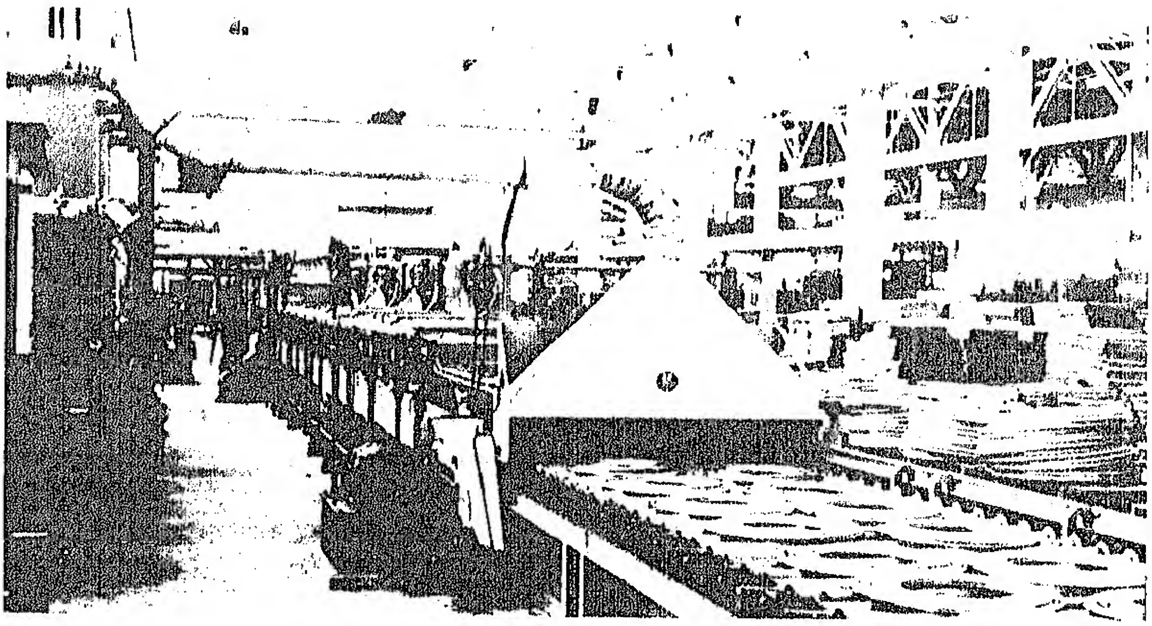


Figure II-34 Single-deck, roller-hearth, wire-annealing furnace

vigorously in the furnace by a fan, or where circulating fans are arranged below the arch of a direct-fired furnace, with suitable alloy baffles to provide circulating around the metal.

Radiant-tube-heated annealing furnaces are usually employed when oil or gas with a high sulfur content is used and competes directly with electrically heated furnaces employing resistor elements. The tubes are made of heat-resisting alloy and distributed in the hearth of the furnace. The firing is done at one end, and the products of combustion leaving the other end are carried away by arrangements of ducts and hoods. The heat from the tubes usually is circulated in the heating chamber by fans to cause a high volume of circulated atmosphere around the material being annealed.

Temperature control: Heating or annealing-furnace temperatures are controlled by means of thermocouples connected to a standard control panel, equipped with an automatic indicating and recording instrument.

The heating chambers are generally automatically controlled in two zones. As a rule, there is a recording controlling potentiometer pyrometer in the soaking or finishing zone and an indicating controlling potentiometer in the charging zone. The instruments should be placed where they will be free from excessive high or low temperatures, excessive vibration or shock, and dirt and dust.

3. Seamless Tubes

Tubes are generally produced by two methods - hot piercing or hot extrusion - both of which start with a solid-cast cylindrical billet. The method used to produce tubes by drawing a blank into a shell is not extensively employed. The production of tubes from cast shells has been generally discarded, mainly because of high cost and poor quality.

Preliminary forming by piercing or extrusion is followed by stages of cold drawing. Between the drawing operation, pickling and annealing take place.

The major steps in tube production may be classified as follows:

a. Preliminary forming by
   1. Piercing.
   2. Extrusion.
b. Pointing (before drawing).
c. Cold drawing, annealing and pickling in steps.

d. Straightening by
   1. Roll.
   2. Medart.
   3. Block.
   4. Hand.
e. Finishing.

Piercing

In forming tubes by piercing, the preheated billet is forced over a mandrel, by means of a rolling operation to form a shell. The temperature to which the billet is heated is determined by the composition of the metal and ranges from 1,100° to 1,600° F.

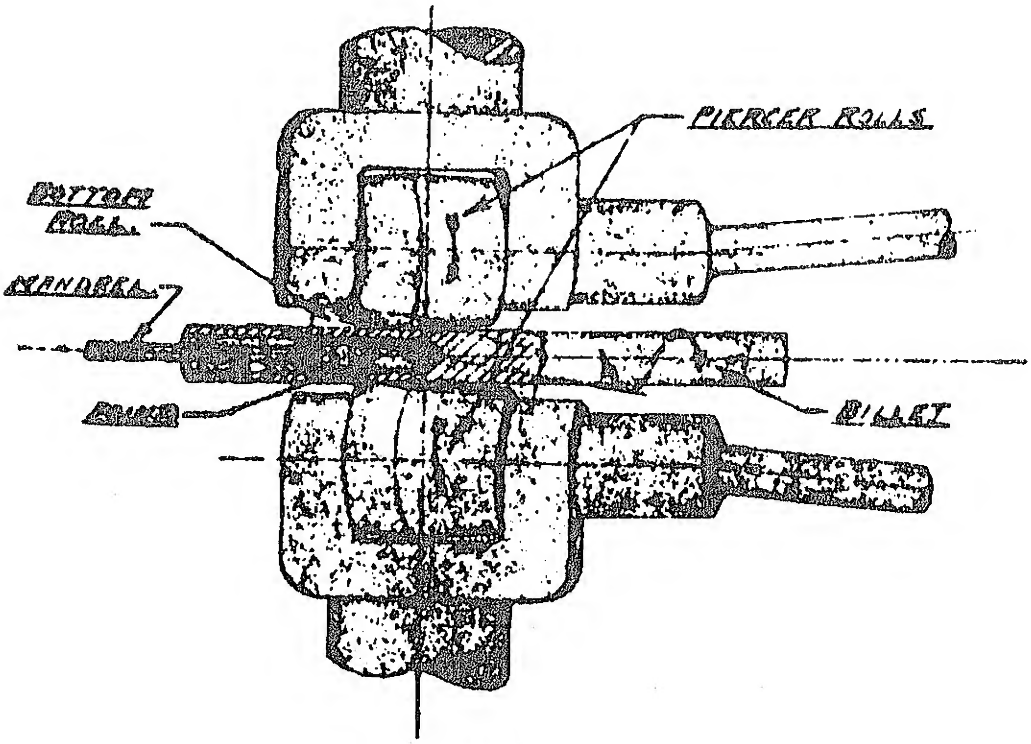
The extrusion operation consists in forcing the preheated billet under compression to pass between a die and a mandrel. The metal is preheated to a temperature sufficient to keep it in a semiplastic state, which requires a range from 1,200° to 2,000° F., depending on the material.

The total cost of making tubes by the piercing method is cheaper than by the extrusion method. The number of billets that can be pierced per hour varies between 60 and 70, requiring 1 or 2 operators, but only about 40 to 50 billets can be extruded per hour requiring 4 to 6 operators. Furthermore, extrusion generates scrap in the form of slugs and butts, as well as the metal extrusion shell, and no such scrap is generated when piercing.

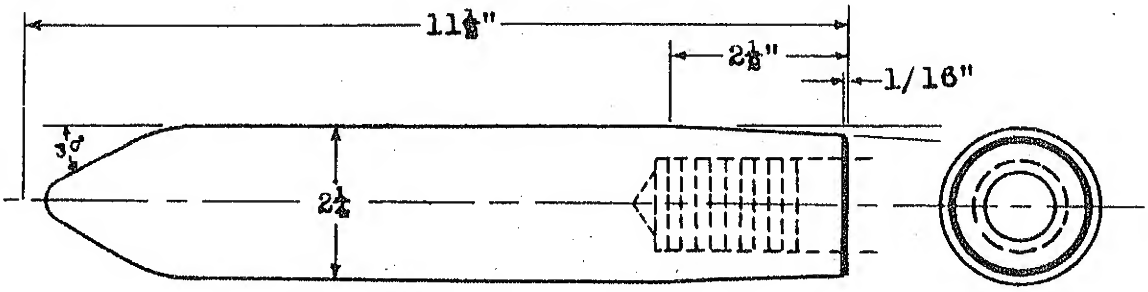
Not all alloys, however, can be pierced. To pierce an alloy, it must be one that has a certain minimum tensile strength, even at red heat. Copper has this qualification, but an alloy of copper and zinc that contains an appreciable amount of lead cannot be pierced, as it has low hot-tensile strength. To illustrate, Admiralty Brass, an alloy used for condenser tubes in connection with salt-water boilers, contains approximately 71 percent of copper, 28 percent zinc, and 1 percent tin. If this were pierced or hot-rolled, it would merely crumble. Various other alloys cannot be pierced, but most of them can be extruded.

Piercing mill: The essential mechanical parts of a piercing mill, commonly known as a "Mannesmann machine", are two main driving rolls, a small guide roll, and a piercing mandrel or arbor.

Figure 35 is a drawing of a three-roll mill and piercer point. The driving rolls are shaped like two sections of a truncated cone, with their bases welded together, forming a ridge in the center of each. The rolls are set at an angle, so that these high points of the roll take the path of a screw thread. When the billet is entered



3 roll piercing mill



Piercer point

Figure II-35 Piercing mill and piercer point

in the rolls and the rolls are rotated, they grip the billet in such a way as to force it ahead through the rolls just as a thread on a screw draws a nut down when the screw is turned.

The guide roll is placed in the opening between the two driving rolls and well below the center line. The center line at the billet as it is being pierced must be below the center line of the main driving rolls, to keep the billet from jumping out through the opening at the top. The guide roll may be straight or ground to a shape somewhat similar to the driven rolls.

The mandrel or arbor is a long bar of steel carrying a high-speed point at one end over which the metal is rolled to form a shell. The other end of the bar is held firmly in a swivel, so that the bar can rotate freely but cannot be withdrawn until the latch that holds it in place is released.

In addition to the above-mentioned essential parts, certain guides are necessary for entering the billet and guiding the shells which leave the rolls. It is quite necessary to keep these guides in first-class condition to prevent scratching or tearing of the surface of the metal, as any such action would produce defects in the surface of the finished tube.

Diameter and length of the rolls are determined by the size of the billet to be pierced. Power requirements are determined by the size and speed of the mill. Two men are required to operate most piercing mills, but some of the newer mills are so arranged that they can be operated by one man. Figure 36 illustrates a tube piercing machine in operation.

Tube-piercing procedure: The size of the billet is determined by the size of shell required for finishing into tubing or pipe.

The size of the finished shell depends upon the size of the mandrel point used and its position in relation to the drive rolls. Generally speaking, the finished shell is approximately of the same outside diameter as the original billet, and the position of the point with regard to the drive roll is regulated to give a certain thickness of wall and the best possible concentricity of gage.

Round billets are received from the plant casting shop or from refiners and generally have an indentation in one end. This indentation provides a starting point for the piercing mandrel and allows the billet to go far enough into the rolls before meeting the resistance of the arbor to allow the rolls to get a good grip.

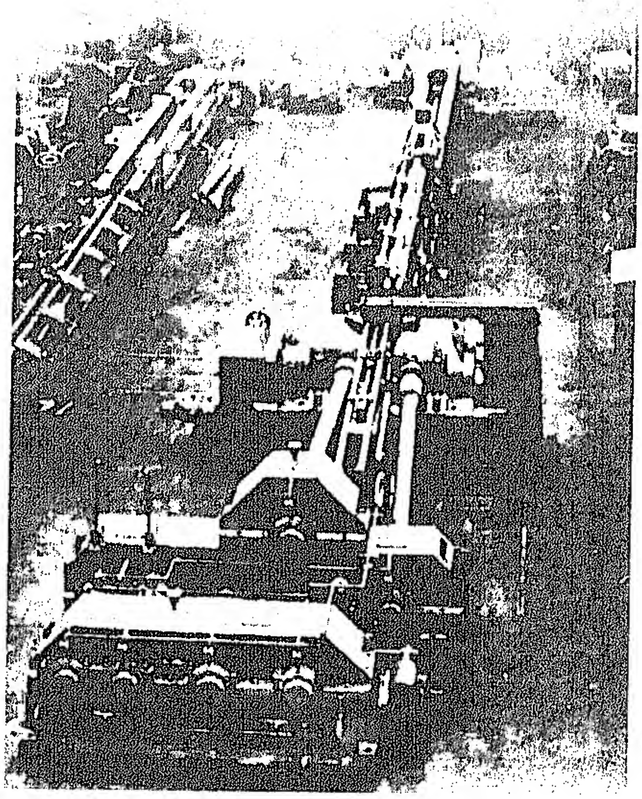


Figure II-36 Piercing mill
(19-inch roll diameter)

The billets are passed through a heating furnace at such a rate as to allow them to be thoroughly heated to a predetermined temperature, after which they roll into a trough, which feeds them one at a time to the driving rolls.

The rolls immediately grip the billet and start it spinning in a rotary manner, at the same time forcing it forward against the point of the mandrel. The mandrel itself is not driven but is free to turn, being supported along its length by so-called napkin rings and at the other end by a swivel joint.

The speed with which the billet goes through the piercing mill depends upon the peripheral speed of the rolls and the amount of pinch taken by the rolls, as measured by the ratio of reduction of the wall thickness at each cycle. If too great a pinch is taken, especially with certain alloys, the inner surface of the shell tends to break and produce defective metal. If too little pinch is taken, the billet may slip badly in the rolls, so that piercing would be impossible.

The power exerted is sufficient to cause the metal to flow over the point of the mandrel; in a fraction of a minute, the whole billet has passed through the rolls over the point of the mandrel, resulting in a finished shell. The mandrel is then withdrawn and the shell rolled out of the machine into a tub of water, where it is quenched. The approximate time of piercing a 3- by 50-inch billet is 26 to 35 seconds.

The pierced shells are cooled in a water tub, then go to saws where the ends are trimmed and are inspected inside and out for visible defects. The shells go through a pickling solution of sulfuric acid and are then pointed, after which they are then put through a series of operations to finish size.

Extrusion

Extrusion is an intermediate operation to rough-form material from the cast state to a form approximating that of a tube, rod, or shape.

Certain alloys can be extruded much more readily than others because of lower resistance to deformation in the hot condition. For example, starting with an alloy containing 55 percent copper, the required extrusion pressures are at their lowest point. As the copper content is increased, the metal remains quite readily workable in the hot condition until an alloy containing approximately 63 percent copper is used, beyond which the required pressure increases rapidly.

It was formerly considered impractical to extrude metal containing 66 percent copper and upward; but now, with increased pressures available and better knowledge of metallurgy, almost any of the copper alloys can be extruded.

Extrusion machines: These are built both vertical and horizontal. The horizontal machines are in general use. In general, the process of extruding tubes, rods, or shapes is similar, but the arrangement of certain parts of the machine is somewhat different. The die mechanism, including the die, is the same; the container and the container liner are essentially the same in principle. The container liner is backed up by the container to provide greater strength. The difference involves the dummy block, the ram and the piston moving through them. For tube extrusion a second piston and ram within the main piston and ram are employed. The inner ram is called a mandrel and passes back and forth in the main ram independently. The dummy block has a hole through the center, through which the mandrel passes.

The ram is approximately 7 inches in diameter and is backed up by a large hydraulic cylinder approximately 30 inches in diameter. The ram acts upon the dummy block, which in turn transmits the pressure to the billet. The ram is slightly smaller than the dummy block. The thin shell of the metal which pushes back around the circumference of the dummy block does not prevent the ram from being withdrawn at the end of the extrusion cycle. The cylinder is moved under hydraulic pressures which are available up to 5,000 pounds per square inch.

Dies and mandrels: The design and quality of dies and mandrels are most important in the extruding process for tubes. The steels used must stand high pressure as well as high temperature without becoming deformed.

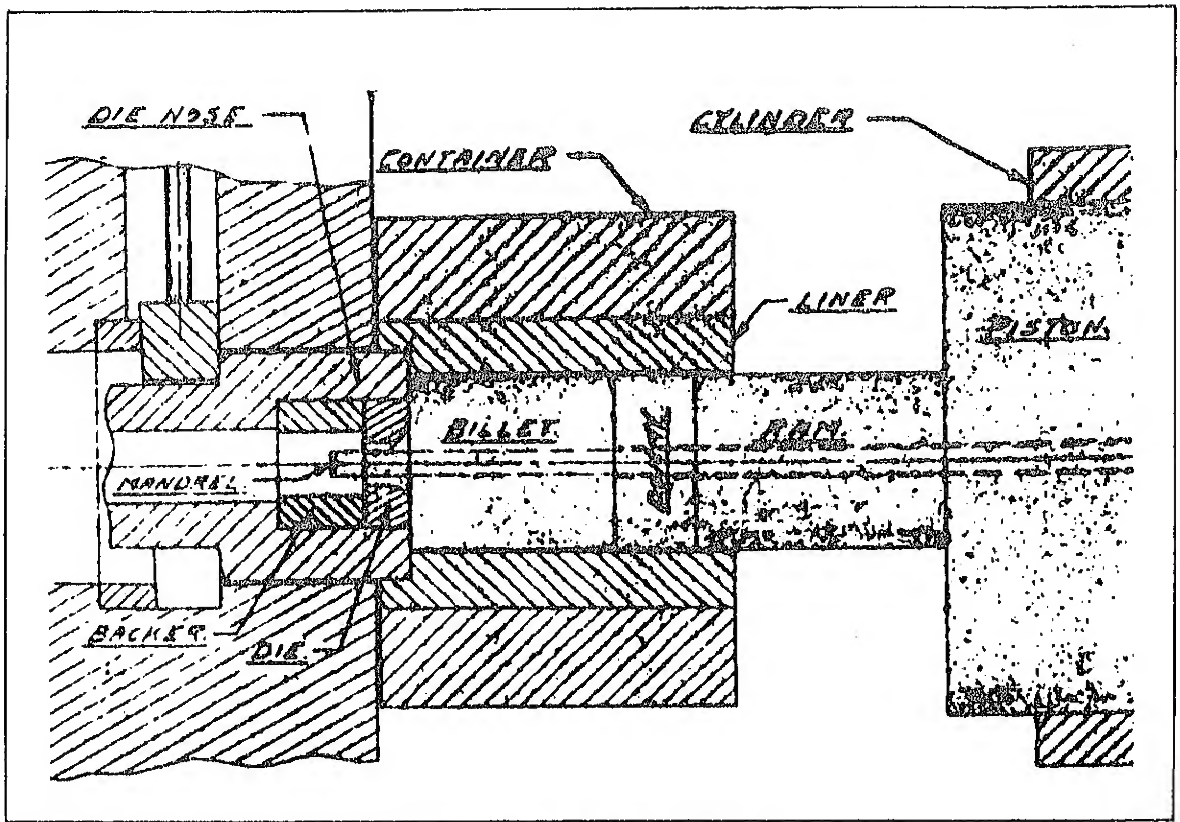
The die is a little over 1 inch in thickness; it would not be able to withstand the pressure applied and is therefore reinforced by means of a backer block, which gives considerable support.

The steel used for mandrels must be able to withstand the impact of cold water from red heat for quick cooling, so that it will be in condition for the next extrusion cycle. It was only after considerable experience and research that proper steels were developed.

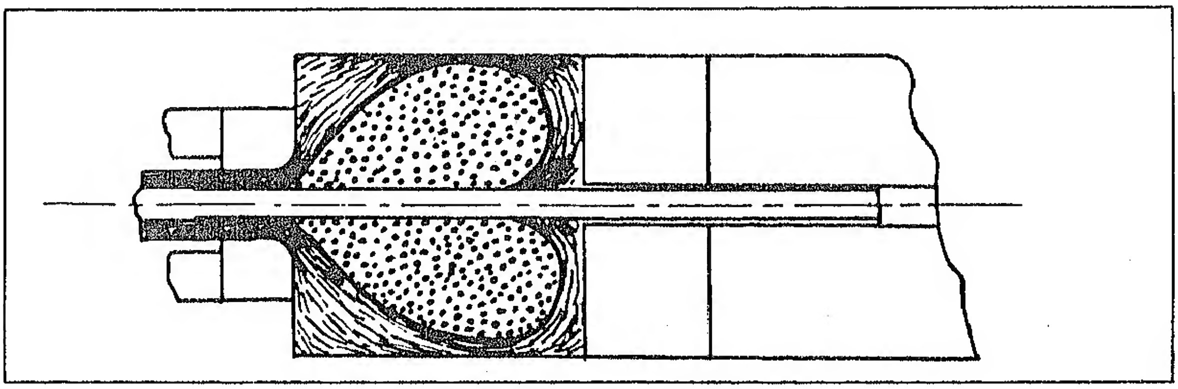
Figure 37 is a drawing showing the essential parts of a horizontal tube extrusion machine and a drawing showing a billet partly extruded. The only difference in the flow of the metal when a tube shell is extruded as compared with rod extruding is that, due to the fact that the arbor extends through the billet, the cooler metal from the outside of the billet cannot go beyond this point of restriction. Furthermore it does not flow in directly along the face of the arbor but remains at least 1/8 inch away from the face of the arbor. The metal directly against the arbor generally is clear.

Extruded tubes: The great advantage of extrusion over other methods of producing tubes is that it lends itself to the production of tubes of various alloys which cannot be pierced and also yields a product that, from the standpoint of freedom from physical defects, compares very favorably with any that can be made by the cast-shell method.

Extruded shells are generally dense and free from defects other than possibly some core at the back end. The greatest weakness in the extrusion process is the difficulty of producing a concentric shell. An efficient extrusion operation should produce a shell with a wall thickness within 5 percent of perfect concentricity, which implies a uniform wall thickness and a true circular cross section. This, however, is an ideal condition that is uncommon.



Tube-extrusion machine



Partly extruded billet

Figure II-37 Tube-extrusion mill and partly extruded billet

Tube-extrusion procedure: The billets for tube extrusion are received from the plant casting shop and vary in length from 9 to 15 inches. After heating to a predetermined temperature, a billet is delivered to the machine and placed in the container. The steps in the extrusion cycle are shown in figure 38. The mandrel, which has been lightly greased, is withdrawn into the ram so that it projects just enough to hold the dummy block and be nearly flush with the face of the dummy block (a). The main ram then travels forward until the dummy block is brought up against the end of the billet, causing it to fill the container liner, making a tight fit (b). The main ram then stops, then the mandrel moves forward through the billet, punching a hole through it and forcing a small slug of metal through the hole in the die ahead of the end of the mandrel (c). As soon as the end of the mandrel reaches a position just beyond the die its rapid forward motion is stopped, and a new movement proceeds (d).

In this movement the main ram progresses forward, forcing the metal out through the circular opening between the die and the mandrel. At the same time that the forward motion of the large ram is taking place, the small mandrel is also moving forward slowly. If the mandrel should stop in its forward movement, the tendency would be for the metal passing through the die to stretch the mandrel, causing it to neck down and be damaged.

The forward movement of the large ram continues up to the point where about 1 inch of butt scrap is left of the original billet. At this point the ram stops, the pressure is relieved, the gate that holds the die head in position is raised, and the ram then proceeds forward, ejecting the butt and die head out of the container. During removal of the butt, the ram is withdrawn (e). An oversize dummy block is then inserted, and the ram makes another forward movement, carrying this oversize dummy through the container, pushing ahead of it the metal shell made by the previous extrusion. If the shell is removed from the container after each extrusion, it is possible to operate with considerable less butt scrap than when the shell is left in the container. Furthermore, the outside surface of the tube is smoother. The ram is again withdrawn, and the mandrel allowed to extend beyond the ram to its full length so that it may be cooled by water quench or spray before the next extrusion.

In the course of the extrusion operation, it is customary to use seven blocks in rotation, so that each block is only used for each seventh billet; therefore, no one block becomes excessively heated. The die is used continuously until it either starts to scratch the product or becomes too small for further use owing to

FIGURE II-38
TUBE-EXTRUSION

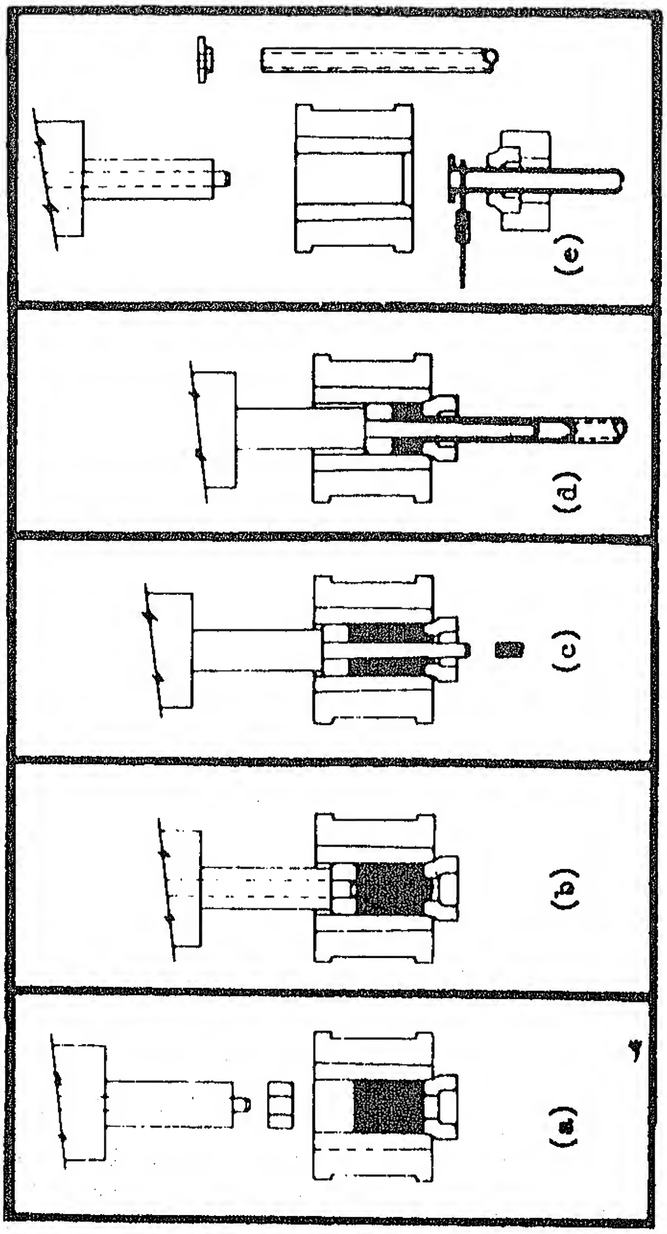


Figure II-38 Steps in tube-extrusion procedure

flow of the steel in the die. This is most noticeable when extrusion is done at generally high temperatures and where the pressures are extremely high.

After the tube has been extruded, it is sprayed or quenched in a tub of water. Then it is pickled, the ends are trimmed off, and it is superficially examined for blisters and core. The front end of the extruded shell or tube is generally off gage, and it is common practice to cut it back or to use it for the point for the subsequent drawing operation.

Pointing

The object of pointing is to prepare tubes for drawing. Pointing is a cold-working operation, and the conical point is about 8 to 10 inches long. The diameter of the point is prescribed by the size of the dies through which the tube must be drawn on the draw benches. A tube is usually pointed to such a diameter that a point can be used for two or more draws, after which the tube is usually too long for redrawing and must be cut into shorter lengths. The previous points are also cut off, as they are usually too large for further use.

The smallest diameter to which a given tube can be pointed depends upon the alloy, temper, outside diameters, and gage. The more ductile an alloy and the softer its temper, the smaller it can be pointed. In some instances, the gage limits the extent to which the tube can be pointed, because as the end of the tube is swaged, its walls thicken, and providing the alloy and temper will allow, a point diameter will be reached that will close up that end of the tube. Pointing the tube further is liable to overtax the machines and may yield points that will break during subsequent drawing operations.

After the tubes have been properly pointed, they are cold-drawn on bull blocks or draw benches. Tubes cold-worked on tube-reducing machines do not require pointing.

Pointing machines: There are three general types of pointing machines in use - squeeze pointers, rotary pointers, and press pointers.

The squeeze pointer (fig. 39) is generally used for pointing pierced or extruded shells and for large-diameter tubes.

The rotary pointer, whose opposing dies rotate about the center of the tube, swage the end of the tube as it is manually pushed into the die hole. This type of pointer is not adapted to pointing heavy tubing. Figure 40 illustrates a rotary pointer in operation.

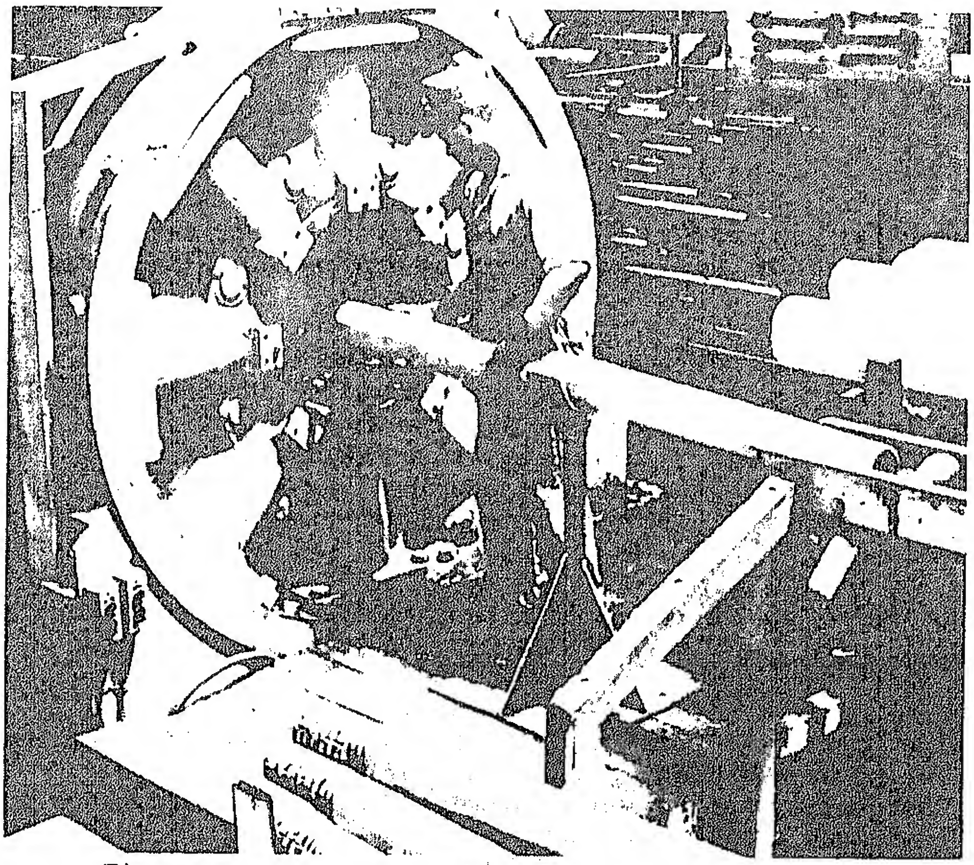


Figure II-39 Squeeze pointer

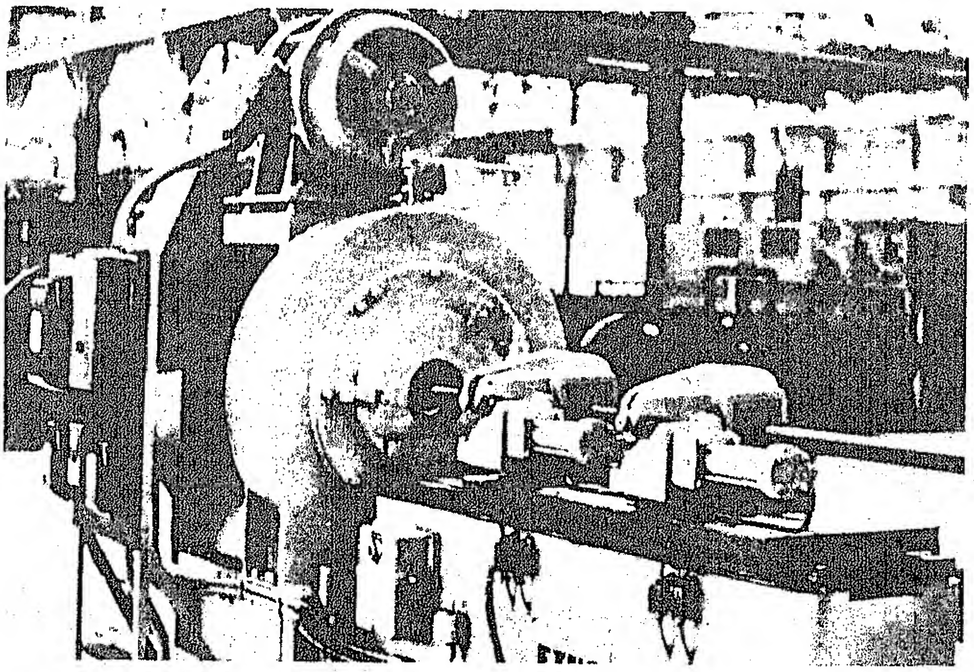


Figure II-40 Rotary pointer

223535 O—52——12

The press pointer utilizes a set of cast-iron dies. The dies oppose each other in motion on a vertical plane, swaging the end of the tube to a conical point as it is manually forced and slowly rotated into the various holes, which may number from 2 to 8.

Figure 41 illustrates various pointing dies and a typical point.

Drawing

Whether the preliminary forming was by piercing or extrusion, tubes are finished to gage by cold working and annealing in a number of stages.

Cold drawing reduces the outside and inside diameters and at the same time reduces the wall thickness and circular area of the tube. The inside diameter, as a rule, is reduced slightly less than the outside diameter, and the operation results in a smooth-finished surface inside and outside. During this operation the metal becomes hardened; hence, at intermediate stages the tubes are annealed.

The number of draws required to process a tube from its extruded or pierced size to its finished size depends upon the following factors:

1. The power and "pointing" capacities of the pointing machines.
2. The power and drawing capacities of the draw benches.
3. The quality and design of drawing dies and drawing plugs or mandrels available.
4. The effectiveness of available tube-drawing lubricants.
5. The maximum reduction in cross-sectional area that the alloy will allow in one draw in a soft condition, and the maximum reduction in cross-sectional area that the alloy will allow on a subsequent redraw in a hard condition.
6. Surface condition of the tubes, whether clean, etc.

Tube drawing to finished size on draw benches is normally accomplished by one of three methods: (1) Drawing over a fixed mandrel or "plug"; (2) drawing by sinking through a die with no mandrel on the inside; and (3) drawing on and with a mandrel on the inside, in which method the mandrel or arbor travels through the die with the tube.

Tubes are also finished to size on bull blocks and on tube-reducing machines. Bull blocks are being employed more extensively owing to the demand for long, coiled lengths.

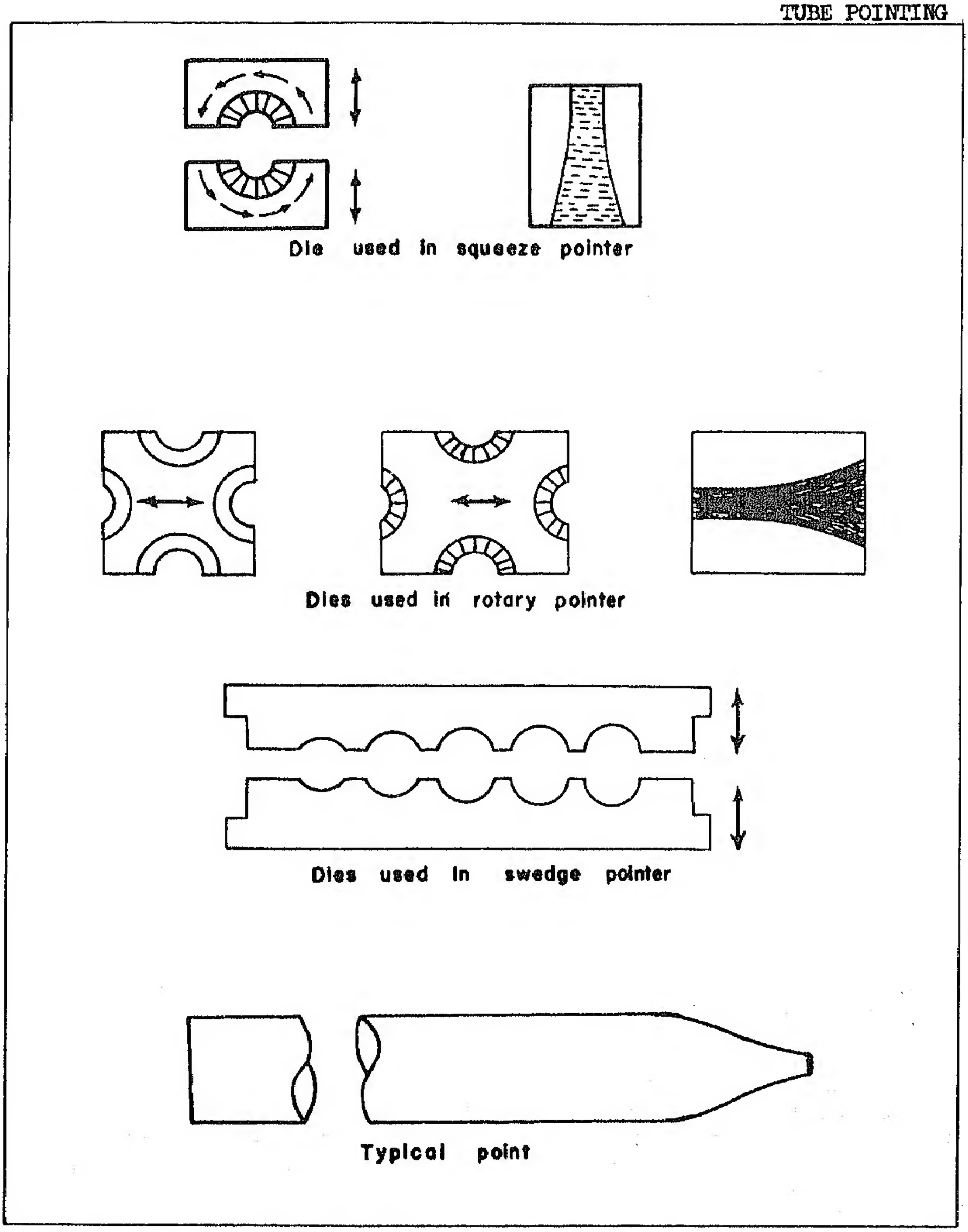


Figure II-41 Dies used in tube pointing and typical plant

The tube-reducing machine employs dies with tapered grooves. The dies are rocked back and forth over the tube, compressing the metal of the tube against a mandrel which governs the inside diameter. The tube is fed through the dies intermittently and rotated so as to distribute the working action over the entire circumference of the tube.

The drawing cycle of the draw bench is as follows:

1. The tube is threaded, tail first, over the pin and rod, which are anchored in the back end of the bench.
2. The pointed end of the tube is then placed through the die, where it is grasped by the jaws of a carriage.
3. The carriage pulls the tube through the die over the pin. The size and shape of the pin determine the inside diameter of the tube; and the size and shape of the opening in the die determine the outside diameter of the finished tube.

Figure 42 shows drawing of: (1) The pointed tube; (2) pin; (3) die; (4) pin, tube and die assembly; (5) tube partly drawn through die; and (6) a complete draw bench.

Draw bench: The draw bench consists of a horizontal frame that has a mechanical drive at one end, and, at the opposite end, a die through which the tube is drawn. An endless, square-linked chain passes through the center, lying in a channel at the top of the frame and returning beneath the bench, and over an idler at the opposite end of the bench. The chain sprocket is driven by a variable-speed motor. A carriage equipped with jaws to grip the pointed end of the tube runs on tracks along the top of the draw bench and is automatically engaged with the continuous chain. Modern draw benches are constructed to draw one or more tubes simultaneously.

The benches are rated in pounds pull, corresponding to the rated capacity of the main chain and the die stand. Common ratings are 1,000 up to 300,000 pounds, although some benches are rated as low as 500 pounds.

The smaller benches are used for drawing capillary tubing or items even as small as hypodermic-needle stock, while at the other extreme the larger benches are used to work tubes up to 16 inches in diameter.

The output of modern draw benches is as much as two to three times the output of most of the benches in use 10 years ago, owing to increased chain speeds, large driving motors, heavier reductions, high-speed carriage return, and improved handling equipment.

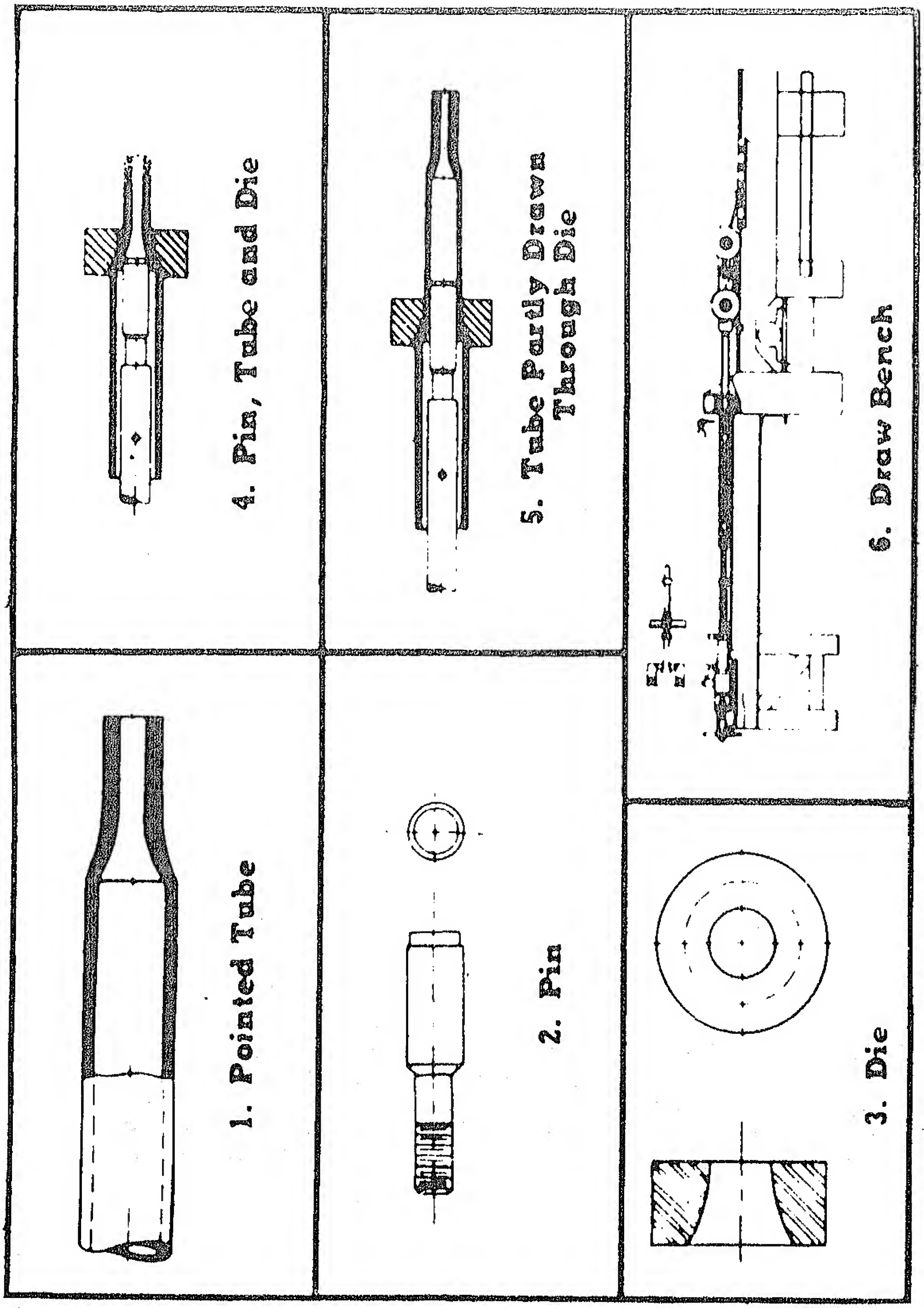


Figure II-42 Draw bench and draw bench assembly

Figure 43 illustrates a draw bench in operation, drawing three tubes.

Figure 44 illustrates a 5,000-pound rack-type draw bench, maximum length 55 feet. Draw speeds, up to 300 r.p.m.

Figure 45 illustrates a 48-inch-diameter bull block for drawing long-length copper tubing from 1-inch to 1 1/8-inch O. D. The block is equipped with pay-off tray, snap shears, pusher, and pointer.

Tube Straightening

Straightening is done after drawing to finish size or after the last anneal, to eliminate any general or local curvature resulting from mill processing.

Straightening may be done by one of four methods, depending upon the size and temper of the finished tube.

Roll straightening: This is done in a machine equipped with 8, 12, 16, and 20 or more rolls, with a semicircular circumferential groove to fit the size of the tube to be straightened. The rolls are arranged in tandem and staggered and adjusted so that, as the tube is passed between them, it is slightly sprung back and forth by the rolls. Each set of eight or more rolls is divided into two groups. One group springs the tube back and forth in a horizontal plane and the other group in a vertical plane. Of each group, only one side is power-driven; the opposite revolve freely. With proper adjustments, any hooks, bows, and crooks can be straightened.

Medart straightening: This uses two power-driven rolls, one with a straight surface and the other with a concave surface. The two rolls are contraposed at vertical angles. Between and below the center of the rolls is a babbitt guide. Once the tube is started in the rolls, the machine propels it through while rotating it and effects a straight tube. This method of straightening makes possible the manufacture of copper tubing with a satisfactory spiral bending temper. The tubes are annealed before Medarting, which in turn hardens and stiffens the annealed tube. The machine also imparts to the tube a peculiar finish resembling a polish.

Block straightening: This is a manual operation performed on tubing that cannot be rolled or Medart-straightened because lengths are too short, diameters too large, or gages too thin. Also, any tubing that may need just a slight crook removed can be very conveniently block-straightened. One end of the tube is anchored and the remaining part of the tube sprung to a degree that will remove any bends.

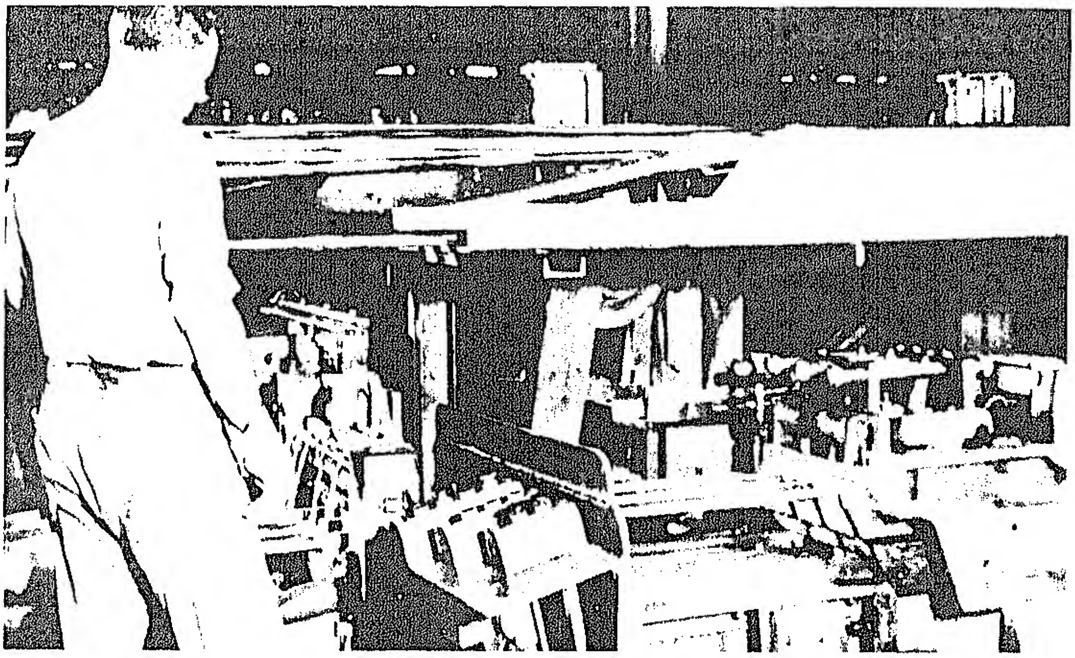


Figure II-43 Single-chain draw bench

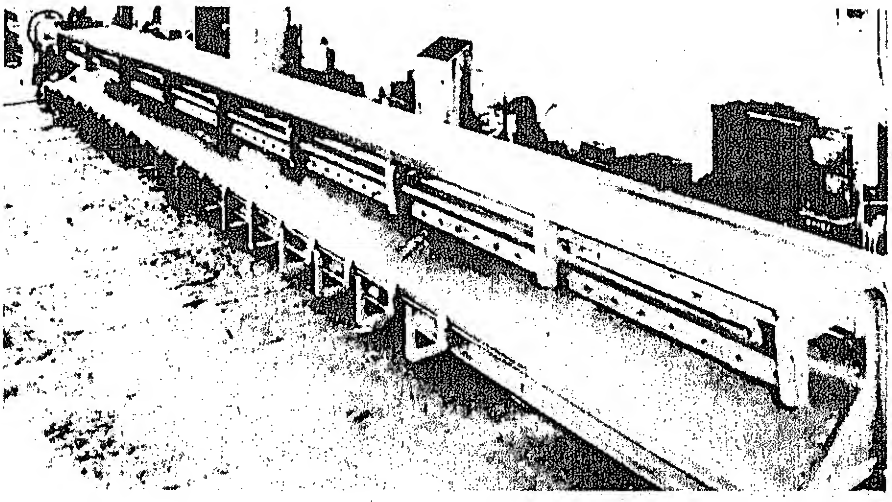


Figure II-44 Rack-type draw bench

FIGURE II-45
BULL BLOCK

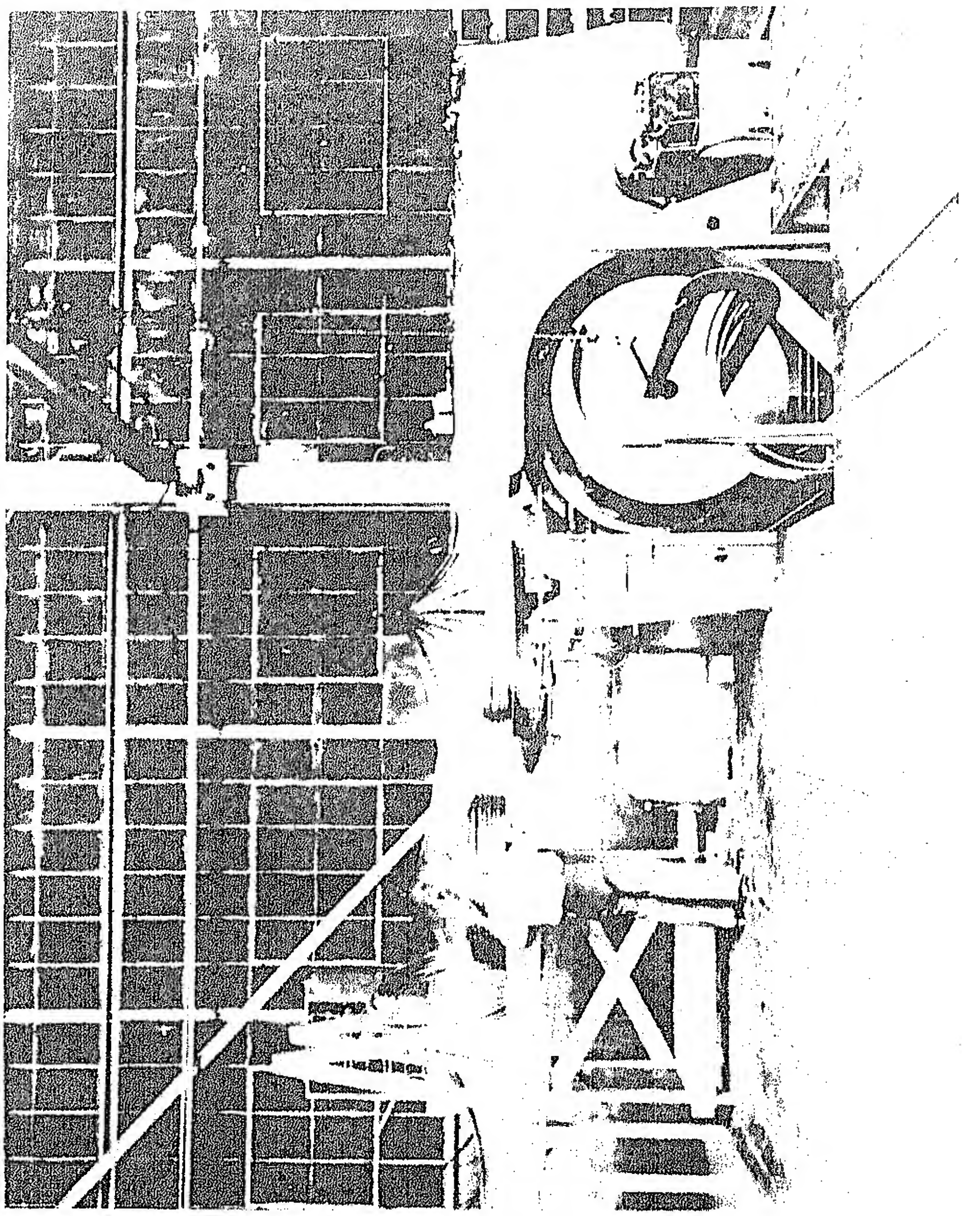


Figure II-45 48-inch-diameter bull block, drawing copper tubing

Hand straightening: This is a manual method ordinarily used for straightening tubes over 3.500 inches in diameter. One end of the tube is rested upon a low horse, and the other end is raised above the floor to a height that will spring the tube straight when allowed to drop or be forcefully thrown down. Tubes are also straightened by placing the ends on blocks and moving a hand-operated hydraulic press along the length of the tube.

After straightening, the finished tubes are cut to specific length and furnished in coils or straight lengths.

4. Rods and Shapes

Copper-base alloy rod is generally produced by the extrusion method and processed to finish by cold working. Rods approximately 2 inches in diameter and larger usually are hot-rolled in mills equipped with grooved rolls to finish size, the rough surface is grounded or turned on a lathe. In some instances rods are cold-rolled from cast billets to finish.

Brass rod produced by the extrusion method is generally known as "free-turning" or "screw-machine" rod. The alloy for this purpose usually contains $2\frac{1}{2}$ to $3\frac{1}{2}$ percent lead and 60 to 62 percent copper, the balance zinc. Other mixtures are chosen with reference to ultimate use of the product, but most of them are processed the same as free-turning rod.

Extruded rods are round, hexagonal, or square cross sections. Rods as small as 9/32 inch in diameter may be extruded in particularly soft alloys, but standard practice is not to extrude rods smaller than 7/16 or 1/2 inch in diameter.

Rod to be finished 3/4 inch in diameter and smaller is usually extruded through a multiple-hole die, and is generally coiled hot as it comes out of the extruder. Larger diameters are extruded through a one-hole die and in straight lengths.

Shapes consist of a variety of simple and complex cross sections and sizes. Extrusion is often the only economical production method, as it produces shapes with good surface, freedom from porosity, uniform properties, and good machinability. Extrusion subjects the metal to high pressures and a thorough working, which develops a dense, fine-grained structure. Successful extrusion of complex shapes depends mainly on die design. Rod dies are similar to tube dies and are subjected to the same condition.

The major steps in rod or shape production may be classified as follows: (1) Extrusion; (2) pointing; (3) cold drawing, pickling and annealing in steps; (4) finishing.

Extrusion

Extrusion machines are similar to the machines used to fabricate tube, except for use of a solid ram and dummy block and elimination of a mandrel. The operation is simpler and has a shorter extrusion time cycle.

Figure 46 shows a sectional view of a typical rod extrusion machine and partly extruded billet.

The extrusion operation is essentially the same as for tube extrusion. The billets are received from the plant-casting shop and vary in diameter and length, depending on the alloy and the size and number of pieces to be extruded. The rods and shapes being solid, it is unnecessary to have two movements of the ram, as in the case of tube extrusion.

As the pressure is exerted against the billet, there is obviously no place for the metal to go except out through the hole in the die, and it therefore flows through this hole, forming a rod or shape having the exact contour of the die opening except for cooling shrinkage for which allowance is made in die design.

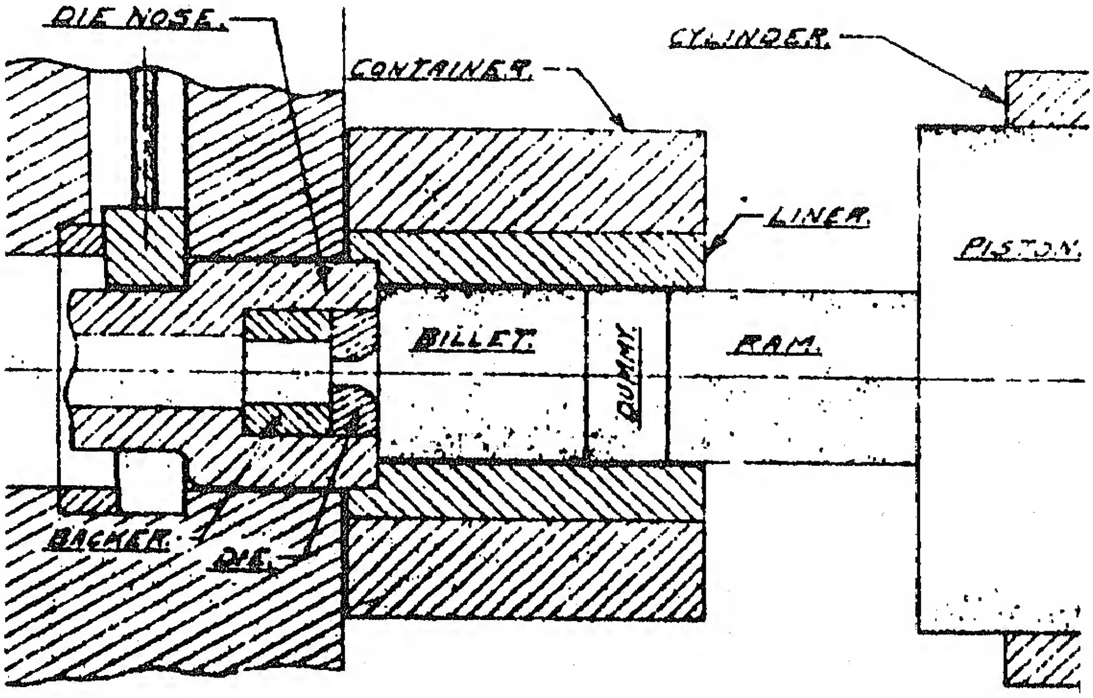
In the early stage of extrusion, the heart of the billet seems to funnel down through the die operation, but after the billet is about a third of the way through the die, the metal starts flowing from the outside shell of the billet across the face of the dummy block and turning in at the center of the billet. A cross-section of any billet taken when the extrusion is partially completed, if polished and etched, will show this typical flow of material.

In general practice, at the end of the extrusion operation, a butt of 2 or 3 inches is left. In order to make certain that the rod after being extruded does not contain core for a part of its length, all rods are given a nick and break test, so that the fractured surface can be examined for flaws. A saw cut will not disclose a core defect.

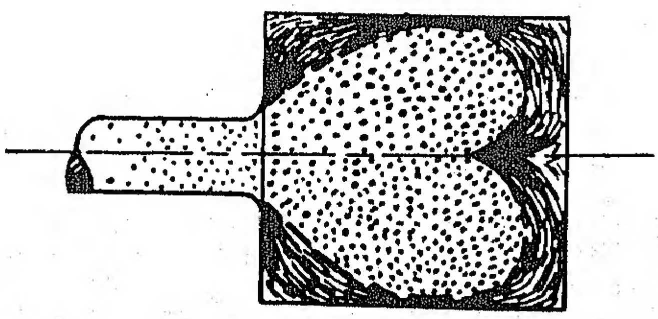
The extruded rod or shape after leaving the extrusion machine carries a light oxide film which is readily removed by pickling.

Pointing

All rods and shapes are pointed before receiving a drawing operation. The pointing is accomplished in a swaging machine,



Rod or shape extrusion machine



Partially extruded billet

Figure II-46 Rod extrusion mill and partly extruded billet

by machining, or rolling, or by a combination of these procedures. The point must be small enough in diameter so that it may be inserted through the drawing die.

Drawing

Rods and shapes are brought to final dimensions, or to form a shape that cannot be extruded to the finished form, by one or more cold drawings or sizing operations with suitable intermediate annealing and pickling stages when necessary. The final temper of the rod is controlled by the amount of reduction in area and annealing.

Machines: The machines used for drawing rods and shapes to final dimensions are bull blocks, draw benches, or continuous rod machines.

Bull blocks are generally used for both intermediate and final drawing on small-diameter rod. The block has provisions for a die and a rotating drum on which the material is coiled.

Draw benches used for drawing rods or shapes are similar in design and capacity as described for tubes. A carriage or tongs, having a hook or grip jaw, grips the pointed end and draws the material through the die, reducing it in diameter or changing its form.

Finishing

Rods finished to size on draw benches or bull blocks receive one or all of the following operations: Annealing, straightening, cutting to length, and coiling. Shapes usually are straightened and cut to length after drawing or sizing. The machines and processes used are similar to those described for tube finishing.

Continuous Rod Machine

A continuous rod machine consists of a preliminary straightening unit, drawing unit, cutting-off device, and final straightening and polishing unit.

It has definite advantages over the draw bench, as it automatically reduces the cuts to a specified length in one operation and only one operator is required.

The machine was invented in Europe and was first used in this country in the middle of 1930's. They are built in three sizes and are known as the Schumag continuous-rod machine. Figure 47 illustrates a No. 2 machine, which is designed for round rods from 5/8- to 1-inch diameter and profile rods of corresponding cross-sectional area.

FIGURE II-47
ROD MACHINE

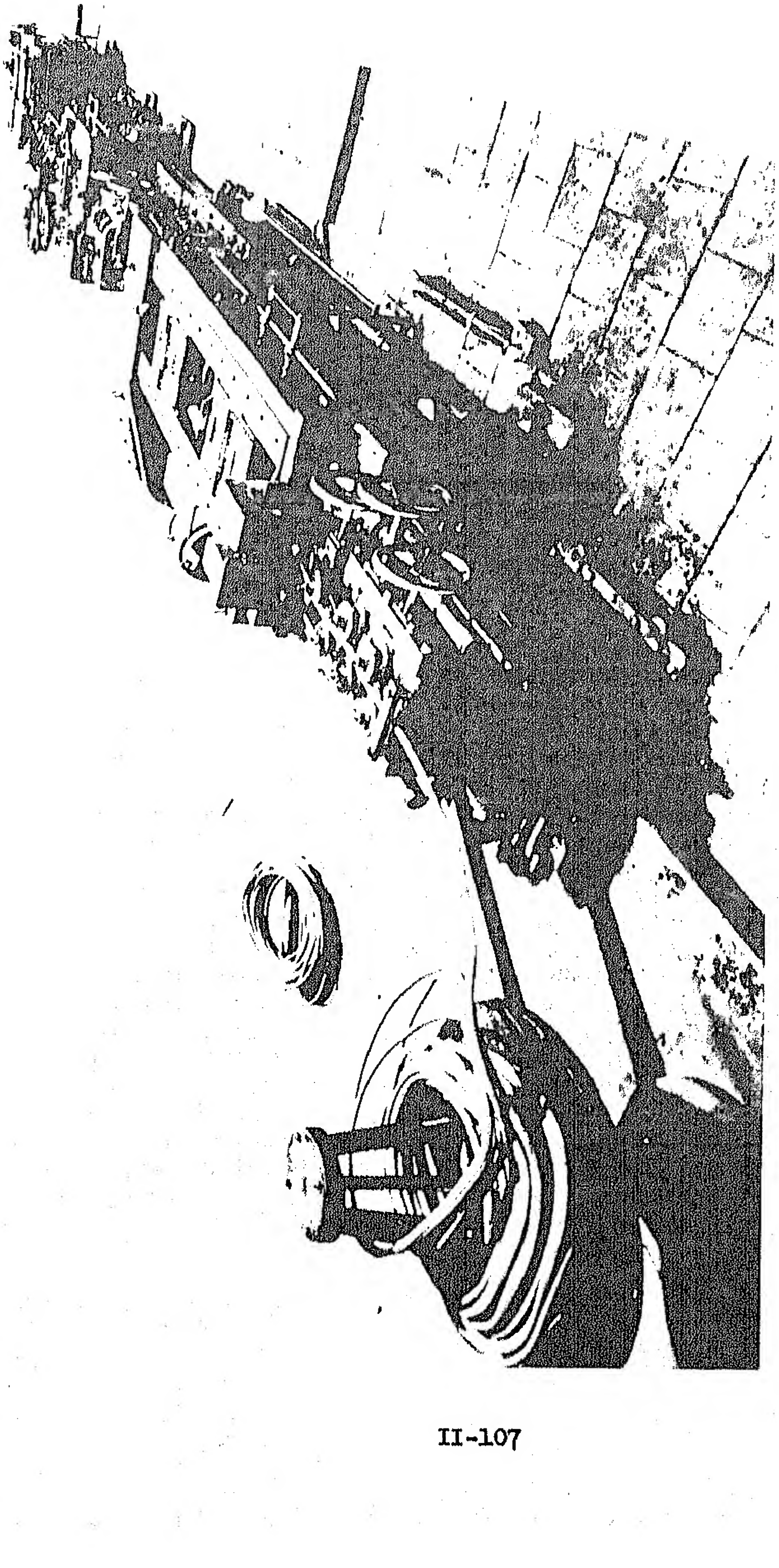


Figure II-47 Continuous rod machine

5. Rolling: Strip, Sheets and Plates

Rolling is the process used to reduce the cross section of a casting by means of compression between two cylindrical rotating bodies, known as rolls. By successive rolling operations, ductile metal can be reduced to an almost unlimited extent, provided that the ductility of the metal is not destroyed by work-hardening. In fabricating copper and copper-base alloys, rolling is applied principally to flat shapes, including strip, sheet, and plates.

The classification of flat-rolled products varies somewhat in the trade; but, in general, flat metal up to 20 inches in width is referred to as strip, metal wider than 20 inches is classified as sheets, and metal thicker than 3/16 or 1/4 inch is called plates. Rolled products of less than 0.006 inch thickness are termed foil.

Rolling may be either hot or cold, depending on conditions. Rolling mills are designated as 2-high, 3-high, or 4-high, according to the number of rolls.

The actual rolling is conducted in a number of stages to progressively reduce the metal thickness. Annealing, milling of the surface of alloys after the first rolling cycle, pickling, and edge trimming may be necessary at various points.

Hot Rolling

There is no essential difference in principle between hot and cold rolling, but in hot rolling advantage is taken of the fact that certain metals and alloys become more malleable at elevated temperature. Copper, for example, is extremely malleable at temperatures between 1,200° and 1,700° F. Maximum reductions per pass are limited by the diameter of the rolls and the horsepower applied. In the hot rolling of copper about 90 percent total reduction is usually taken in 11 or 13 passes, that is, from 4 5/8 inches to about 0.200 inch. With brass alloys that may be hot-rolled, the total reductions and number of passes vary at different plants, depending to a large extent on the thickness of the original casting, which will vary from 3 to 5 inches. It is common with brass alloys to hot roll down to about 0.5 inch for convenience in subsequent surface-milling operations, which become slow and expensive if the gage is too light. Provided the metal or alloy is ductile and malleable while hot, the only limiting factor covering the total possible reduction is the ability to retain the internal heat of the slab, as this heat is constantly being lost by radiation and transmission to the rolls and run-out tables during hot rolling.

Copper-zinc alloys containing 65 percent or more copper can be successfully hot-rolled, provided certain impurities are held to small traces. Lead is the most harmful of the common impurities in its effect on hot-rolling these alloys, owing to the fact that it precipitates at the grain boundaries during solidification after casting and at the normal rolling temperature of 1,300° to 1,600° F. is present in the molten state, thus markedly lowering the cohesion of the grains. Lead should, therefore, preferably be restricted in these alloys to not over 0.03 percent. Below 65 percent copper, lead can be permitted in gradually increasing quantities up to 2 or 3 percent in alloys of 58 to 60 percent copper without harmfully affecting the hot working qualities, due to the presence of the highly plastic beta phase in the alloy. Copper-aluminum, copper-tin, copper-silicon, and other combinations of these alloys can also be successfully hot-rolled, provided harmful impurities are kept to a minimum.

The physical structure of the cast slabs is also important in hot rolling. A long, columnar, grain structure produced by high-temperature pouring and slow cooling is undesirable, as cohesion of this structure is less than in the more equiaxed type produced by lower pouring temperatures and more rapid cooling. A columnar structure tends to produce inter-crystalline fissures, which develop into surface cracks as the structure is changed from a vertical to a horizontal position during the rolling operation.

The harmful effects of structure are most-pronounced during the first three or four passes; once recrystallization takes place, the slabs become more homogeneous and malleable.

From the above considerations, it is obvious that applications of hot rolling are limited to: (1) Specific metals and alloys; (2) correct internal structure; (3) large masses which will retain heat over a period long enough to permit reduction to the required gage, which can be brought about by the use of large castings or high-speed rolling on tandem mills.

Due to the tendency for slabs to spread in width during hot rolling, with resultant stresses on the edges, it is desirable to use edging rolls in conjunction with the hot mill. These edging rolls serve the double purpose of maintaining accurate width and of working the edge structure to prevent edge cracking.

The principal advantages of hot rolling may be summarized as follows: (1) Less power consumption for equivalent reduction; (2) heavier reductions per pass; (3) greater total reduction before annealing is necessary, due to self-annealing properties of the hot metal; (4) flexibility in width of casting due to ability to cross-

roll; (5) faster flow of metal down to milling gage for quick delivery.

Cold Rolling

In contrast to the limited application of hot rolling to alloys of closely controlled analysis, it is possible to cold-roll almost any brass or bronze alloy which has any degree of malleability, provided the reductions are controlled within the limits of malleability of each particular alloy. Cold rolling, therefore, is not only more suitable to a wider variety of alloys, but, in addition, it finds great application in high-speed rolling of strip products.

The ability of metal to be cold-worked depends largely on its mechanical properties, principally tensile strength, hardness, and ductility. Ductility is particularly necessary, since without it no cold work can be done, but strength is also an important factor. Ductility is highest and strength lowest in fully annealed metal; therefore, such material has the greatest capacity for cold working. The effect of cold working is to increase the tensile strength and elastic limit and to reduce the elongation and reduction of area.

When any metal or alloy is subjected to deformation at low temperature below the recrystallization temperature, it becomes more resistant to flow, and with increasing deformation a point is finally reached where further working produces brittleness and it begins to crack. In cold rolling, the rate of deformation is frequently very high, and cracking first appears at the edges of the strip, the zone of least support. These cracks form along planes at a 45° angle to the direction of rolling and indicate that the metal fails under shearing stresses. When this state is reached, the internal structure of the crystal grains has been completely altered and is in the form of long "fibers." These fibers differ in appearance, depending on the structure of the bar before the cold rolling and on the type of alloy. If the structure before rolling was in the cast condition, some indication of the original coarse structure will be present unless the reduction has been considerable (at least 50 percent). If the bar has already been rolled and annealed, the initial structure would be much finer and consequently more readily obliterated after cold rolling. Under microscopic examination at high magnification, it will be seen that the fibrous structure consists of greatly elongated crystals which have been caused to slip along cleavage planes. This condition is accompanied by a considerable increase in hardness. This fibrous structure must be converted to a homogeneous equiaxed condition by annealing at a suitable temperature for a given length of time. The material will thus be reduced to its original soft condition ready for further rolling.

While a 50-percent reduction between anneals is not the maximum iount of cold work that can be applied to brass or copper without ·acture, it is common practice on quality products to anneal after ›lling about 50 percent, or 6 B & S gage numbers. Beyond this ›int, directional properties are produced in the materials which ·e not entirely removed by annealing; in addition, very high roll ·essures are required to effect any considerable further reduction ·r pass, so that it becomes more economical to anneal. This con- ·tion applies more especially to brass alloys containing less than ' percent of copper and to refractory bronze alloys. On the other nd, where directional properties are not a factor, as in the case light-gage strip copper in relatively narrow widths, reductions high as 75 percent can be taken between anneals.

Cold rolling can be performed on single 2-high and 4-high mills on tandem mills of various types and combinations. Cold rolling ows to the best advantage in high-speed strip production, and tput speeds as high as 1,000 feet per minute have been obtained mills equipped with automatic blockers, although such speeds e necessarily accompanied with relatively light reductions. oubles are apt to be encountered at such speeds if reductions e much over 5 percent, owing to the difficulty of removing heat reloped in rolls and roll bearings. Present-day experience indi- :es that a speed of 400 feet per minute on copper or brass 0.003 0.010 inch thick can be satisfactorily maintained with a reduc- on of 20 percent per pass with good control of gage and flatness. :tle is to be gained by increasing speed, if this is done at the :rifice of the amount of reduction that can be obtained thereby. rever, with development of better coolants in the form of soluble .s applied through pressure jets, by hydraulic control of roll :ssure operated by automatic gages and by various other improve- its, there is no reason to believe that the limit of rolling speeds been reached.

The principal advantages of cold rolling over hot rolling are: ater variety of alloys that can be rolled; no heating required ore breakdown; less expensive roll maintenance; better gage trol, making for greater precision in subsequent operations.

With first-class equipment in both cases and weighing all tors except purity of alloys, it would appear from cost data t the advantages are with hot rolling on such alloys as are ally suited for either process. In a highly diversified mill ufacturing relatively small tonnages of a large variety of alloys, oubtedly cold rolling would be preferable, owing to its flexi- ity. In a mill producing large tonnages of copper and simple -rollable alloys, the preference would be toward hot rolling.

23535 O—52— 13

In the steel industry, great strides have been made in the past 15 years in cold-rolling steel at high speed in continuous mills. In the brass and copper industry, on the other hand, where the smaller-tonnage output does not justify the enormous investment, the tendency has been toward retention of older methods, changes being largely confined to improvement in existing types of equipment and in handling facilities.

Rolling Operations

Rolling operations are commonly divided into three stages: (1) Breakdown, (2) rundown, and (3) finishing.

Breakdown rolling is the operation of breaking down the coarse, cast structure of the slab or cake to some heavy-stock point where the rundown operation begins. There is no definite line of demarcation between the end of the breakdown and the beginning of the rundown operation; but in brass rolling - either hot or cold - the breakdown operation is usually considered as taking the metal from the cast thickness of 3 to 5 inches in hot rolling or 1 3/4 to 2 1/2 inches in cold-rolling to a gage of about 0.4 to 0.6 inch.

In hot-rolling the metal is not annealed during this rolling stage; in cold-rolling two or three anneals may be necessary. After the brass is rolled to 0.4 or 0.6 inch, approximately 0.010 to 0.015 inch is milled off the top and bottom surface of the slab to remove any of the cast surface remaining after rolling.

Copper is invariably broken down hot and is not milled, as in the case of brass; consequently, the breakdown operation is carried much further, usually to 0.180 or 0.250 gage. However, a certain amount of scale is left on the surface, and it is considered good practice to pickle the slabs after the hot-rolling operation. In some instances, when the metal is to be rolled to finish gage below 0.010, or if the surface requirements are not too exacting, the pickling after hot rolling can be omitted.

Figure 48 illustrates cold breakdown of brass slabs 2 inches thick by 20 inches wide in a 2-high mill and equipment for delivering the cast slab to the mill and moving the material around the mill between passes.

Figure 49 illustrates hot breakdown of 5-inch-thick copper cakes in a 2-high reversing mill equipped with edging rolls.

Figure 50 illustrates hot breakdown of plates in a 2-high mill.

FIGURE II-48
COLD BREAKDOWN

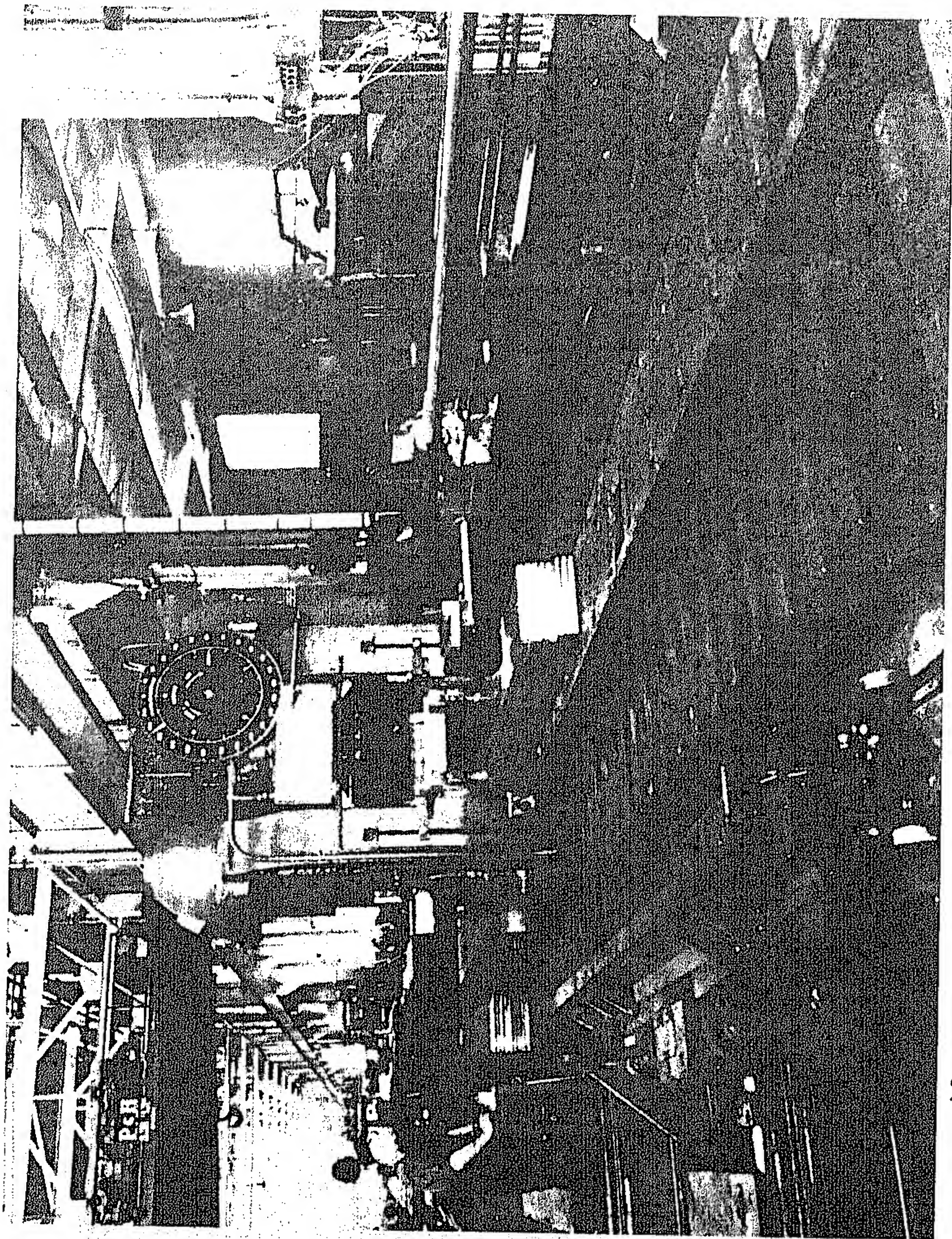


Figure II-48 2-high breakdown mill, cold-rolling brass slabs

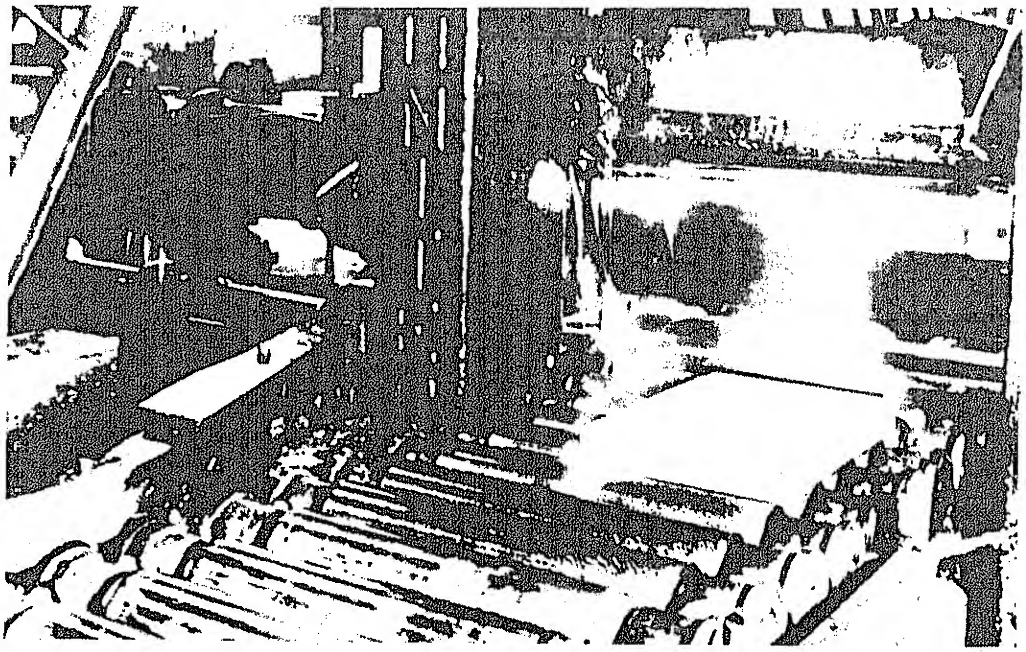


Figure II-49 2-high mill - hot-rolling copper slabs

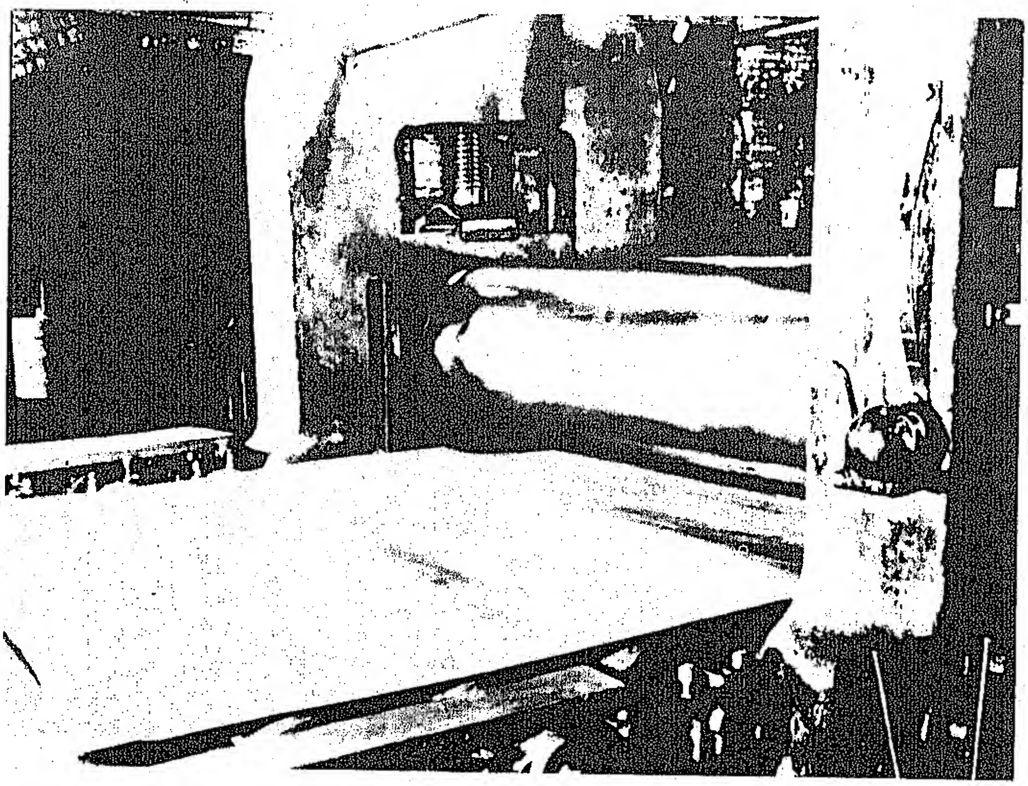


Figure II-50 2-high mill - rolling plates

The 3-high mill, when equipped with three rolls of the same diameter, is generally used for breakdown and in some instances for rundown. The metal receives a reduction when traveling in one direction between the lower and center roll, and in the other direction between the center and upper roll. A lifting table is necessary to raise the metal for the returning pass (reduction) between the center and upper roll. Maintenance costs are high on this type of equipment, and it is being replaced by the 2-high reversing mill for breakdown and the 4-high mill for rundown.

The 3-high mill, when equipped with a small-diameter center roll, is used for rundown and finish rolling.

Contact friction is largely a function of roll diameter and the metal thickness reduction ratio per pass. To reduce friction, it is desirable to use small-diameter rolls. Small rolls, however, will deflect easily under the roll pressures and it is, therefore, necessary to provide stiffness by means of larger-diameter supporting or backing rolls.

It has been found that, as the working rolls are decreased in diameter, the power requirements are lower, and the ratio of reduction may be increased, since it is possible to reduce the metal more between anneals without cracking the edges. This also reduces the number of annealings required.

These considerations led to the development of mills utilizing small diameter work rolls such as the cluster mill and 4-high mills. On account of the high installation and maintenance cost, cluster mills have not become popular in the United States. The first application of 4-high mills for producing sheet copper in the United States was in the late 1920's, and they have since been widely adopted for rundown and finish rolling of strip and sheet.

Four-high mills, as the name implies, consists of four rolls, one on top of another. The middle two are the work rolls, between which the metal is reduced in thickness. The other two are back-up rolls, for the purpose of supporting and lending rigidity to the work rolls. The back-up rolls generally run idle; the work rolls are driven. The work rolls are of considerably smaller diameter than those in an ordinary 2-high mill.

The 3-high mill, incorporating the use of a small work roll, acts on the same principle as the two small working rolls in a 4-high mill, that is, it reduces the area of contact between the roll and metal.

The 3- and 4-high mills are generally used for rolling in one direction only. The initial cost of a 3-high mill is less than that of a 4-high mill.

Rundown rolling is usually done cold, and begins where breakdown rolling leaves off and takes the metal down a number of gages above the finish gage, depending on finishing requirements. Annealing and pickling operations may be introduced at various points during the rundown operations.

Sheets after breakdown in some instances are rolled hot in single sheets, or in packs of two, three, or four sheets together. Plates may be rolled hot or cold.

Figure 51 illustrates cold rundown and finish rolling of heavy-gage strip brass for ammunition.

Finish rolling may consist of several rolling operations from the rundown gage to finish gage. The metal is annealed and pickled at various points between the rolling operations. The rolling terms used to describe the finishing operations between anneals are "ready-to-get-ready", "ready-to-finish", and "finish". The final rolling determines the finish gage and frequently the temper and flatness.

Heating and Annealing

Heating of the cast slabs or cakes before hot rolling and annealing of the material between cold-rolling stages is described in another section, Heating and Annealing.

Pickling

Pickling is generally done after the hot breakdown rolling of sheets and plates. Strip brass is not pickled after breakdown either hot or cold, as the surface of the metal is milled generally when it reaches a gage of 0.4 or 0.6 inch. Brass is generally pickled after each annealing operation. Copper is pickled after the hot breakdown and after annealing unless it is done in a bright annealing furnace.

Rolling Mills

Rolling mills may be 2-, 3-, or 4-high, or specially designed mills, such as the Steckel or the Sendzimir. Diagrams of the roll arrangements of various mills are shown in figure 52.

The earliest and simplest type of rolling mill and the type still very extensively used for all rolling operations especially for finishing is the 2-high mill.

Two-high mills are generally of large diameter, and, owing to the relatively large area of contact between roll and metal, the frictional characteristics are high, with consequent high power requirements.

FIGURE II-51
COLD RUNDOWN

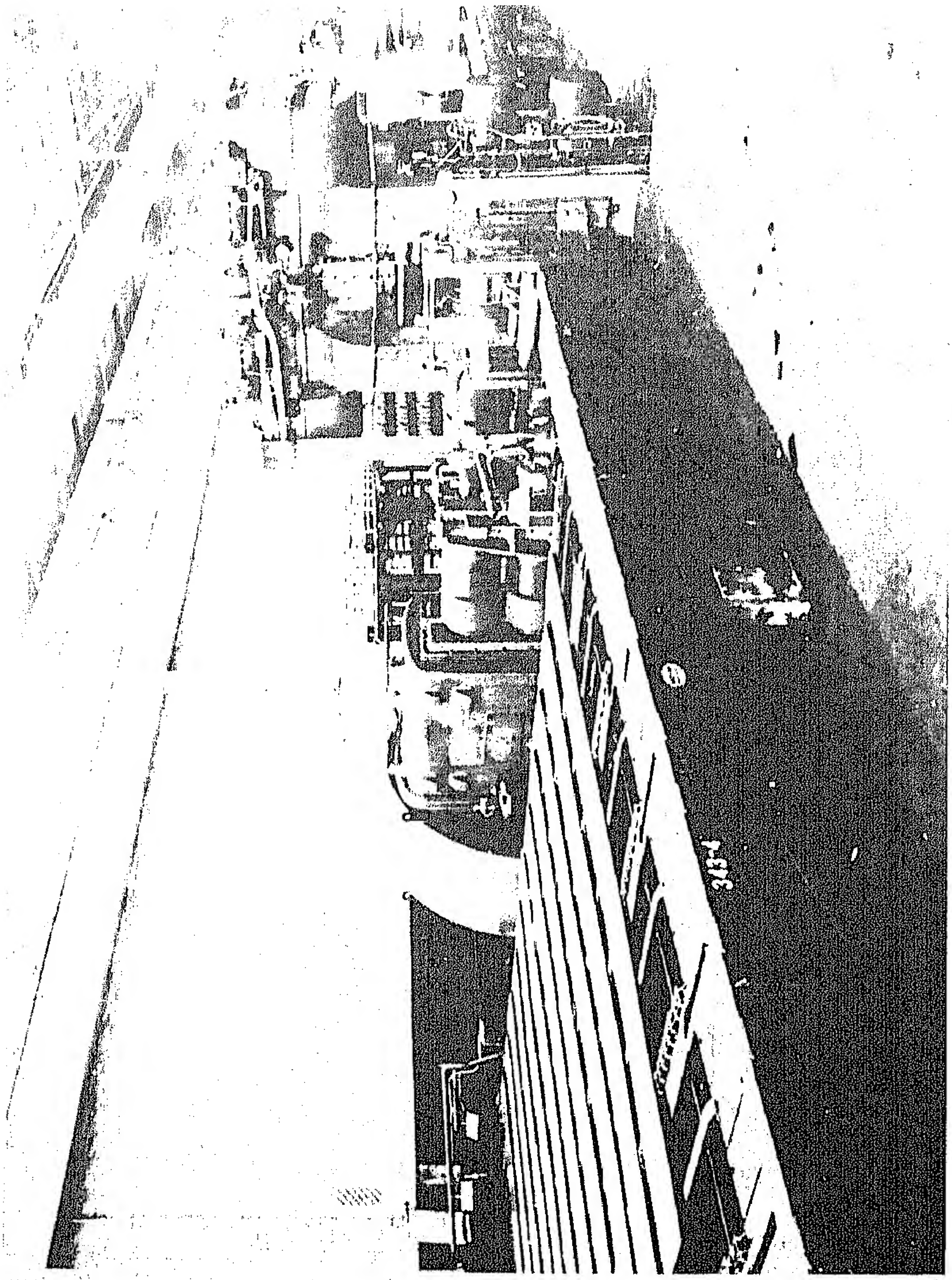


Figure II-51 Cold rundown and finish rolling of heavy brass strip

FIGURE II-52
ARRANGEMENT OF ROLLS

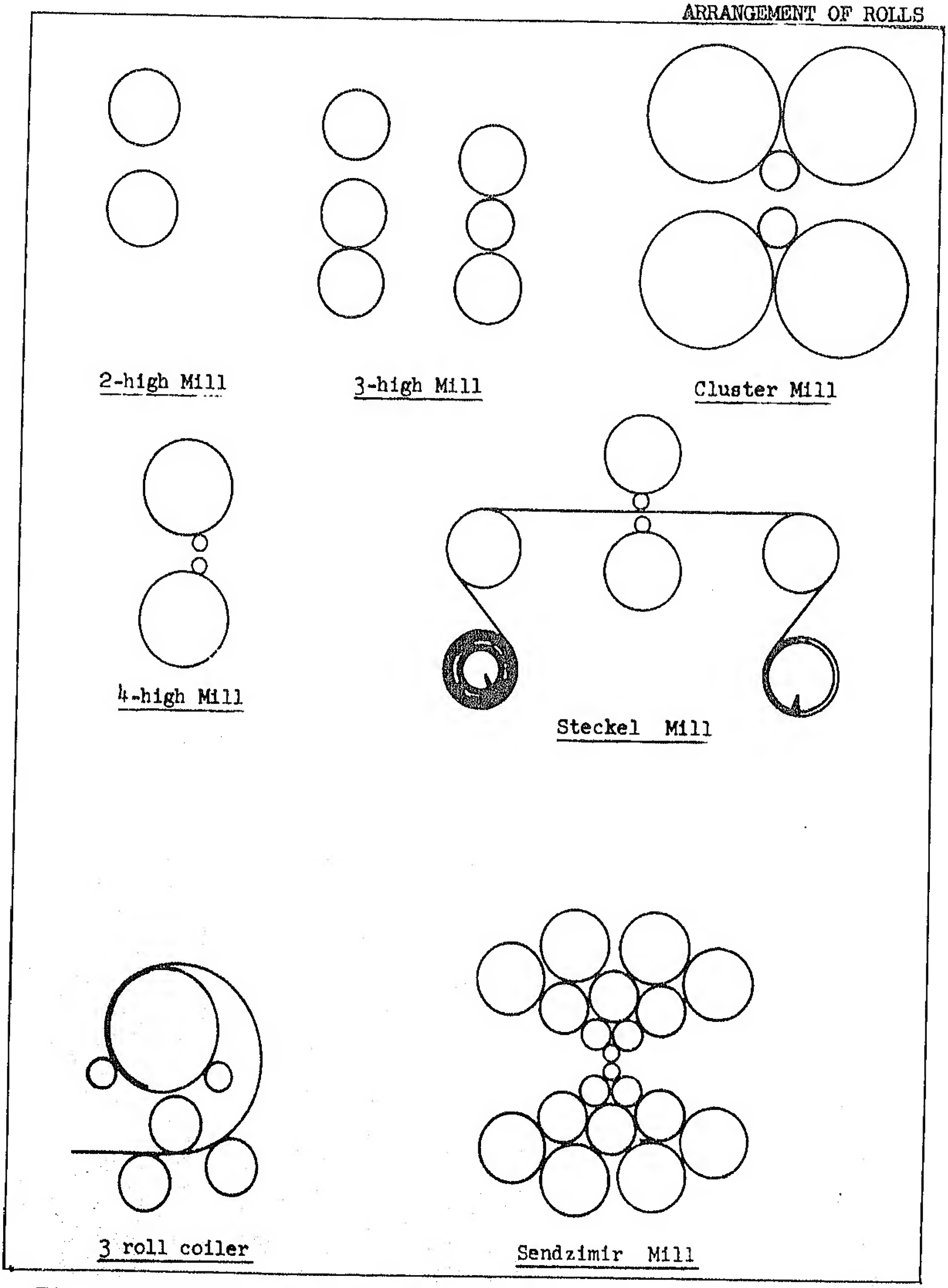


Figure II-52 Arrangement of rolls in 2-high, 3-high, 4-high mills Cluster mills, Steckel mills, 3-roll coiler, and Sendzimir mill

Both the Steckel and Sendzimir mills incorporate the use of small work rolls, in some instances as small as 3/4 inch in diameter. The work rolls in the Steckel mill are backed up in the same manner as in the 4-high mills whereas in the Sendzimir mill they are backed up by a number of rolls.

These mills operate as reversing mills and are equipped with power-driven reels on either side which draw the strip backward and forward through the rolls with a succession of light reductions. The metal remains in the mill until it reaches the required gage. Because the reels must not be permitted to unwind completely, a certain amount of stock must be held on each reel, and since this does not receive the rolling operation the proportion of end scrap is relatively high. To obtain the maximum efficiency extremely long coils of metal must be used.

If coils are not long enough originally, this is accomplished by joining two or more coils into a continuous length by autogenous welding in an inert atmosphere.

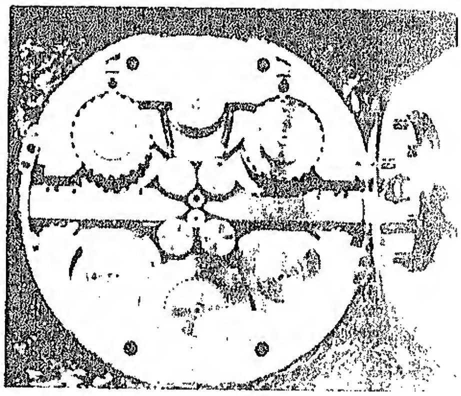
For this reason, Steckel and Sendzimir mills have not been widely adopted in the nonferrous industry and their use is largely confined to light-gage strip steel, rolled in 2,000- to 5,000-pound coils. The Sendzimir mill is of such recent introduction, however, that its ultimate utilization cannot be fully determined.

Operation of these mills is relatively simple, usually requiring one man. The small work rolls stand up surprisingly well between grinds, because the roll separating force is so small. One of the advantages claimed is gage accuracy across the strip and from end to end. Figure 53 shows the roll cradle arrangement and rolls of the Sendzimir mill.

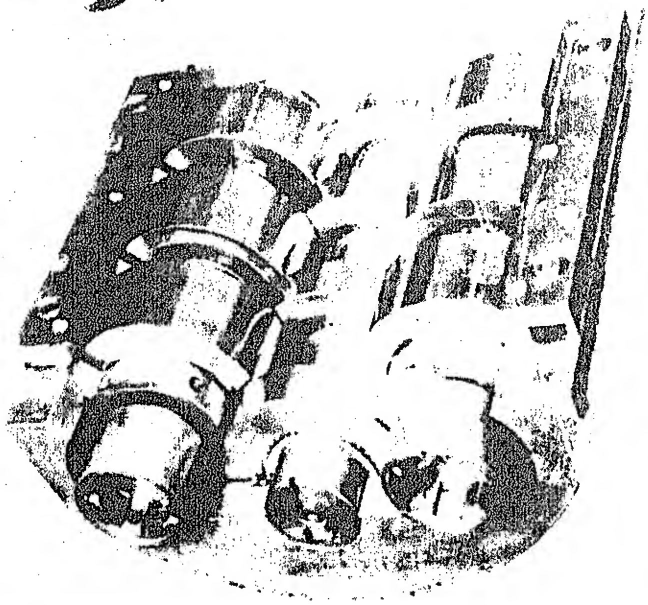
The power requirements in rolling depend to a large extent on the amount of tension, both front and back, applied to the stock during rolling. If a very high tension can be maintained, roll friction can be reduced, with a consequent reduction of power consumption. To carry this principle to its logical conclusion, it is possible to apply enough tension to the discharge side of the mill to enable the stock to be pulled through without applying any power to the rolls themselves. The limiting factor is the tensile strength of the stock, and the reduction must be so adjusted that the pull required is less than the tensile strength of the rolled strip.

<u>Roll materials:</u> The selection of roll materials is based largely on the desired surface hardness, which varies with the alloy and the class of work for which the rolls are designed.

Right: Inside of mill housing, with door open. Note top and bottom cradles, each holding 3 shafts with backing elements, 2 drive rolls and 1 work roll.



Left: View of 1 cradle with rolls in place.



Right: View of same cradle with rolls removed.

Figure II-53 Rollcradle arrangement and rolls of the Sendzimir Mill

For hot rolling, it is common practice to use chilled cast-iron rolls, the roll face having a scleroscope hardness between 60 and 70. Cold breakdown rolls are preferably of cast alloy steel, with a hardness of about 60 scleroscope. Rundown rolls may be of steel or chilled iron but should be harder than breakdown rolls. Scleroscope hardness should run about 70 for chilled iron and 85 for steel.

Finishing rolls, both 2-high and 4-high, require the maximum hardness for preserving the surface and are generally of forged alloy steel with a scleroscope hardness of about 95 or harder.

Roll bearings: The function of bearings in a rolling mill is twofold. They hold the rolls in correct alinement and also transfer pressure to the roll necks and thence to the body of the rolls.

Improvement in rolling-mill efficiency has coincided largely with improvement in design and type of bearing materials. Roller bearings are preferred for the large-diameter backing rolls in 4-high mills; synthetic plastic or composition bearings find their best application for rolling speeds over 120 feet per minute; and flood-lubricated sleeve bearings are especially suited for heavy duty at high speeds.

Auxiliary rolling-mill equipment: In addition to the mill housings, drive, and rolls, the rolling-mill assembly consists of guides and plates for feeding the slabs into the rolls and runout table for the breakdown mill; rundown mills have coilers on the delivery side for metal in coils; sheets and plates are usually rolled in single units and require only runout tables. Finishing mills are generally equipped with blockers or coilers, depending on the gage being finished.

Depending on the thickness of the metal as it is delivered from the rolls, it is either coiled without tension or blocked on a spool, except sheets and plates.

Coiling is generally done on metal heavier than 0.060 inch thickness. The metal leaving the rolls is passed through a three-roll coiler, the rolls being so arranged that they impart a permanent bend in the strip, causing it to roll up on itself and so form a coil as shown in the lower left-hand diagram of figure 52.

In nonferrous rolling it is common practice to coil in an upward direction (overcoiling), whereas in steel mills, because the coils are larger, the practice is to coil downward (undercoiling). With increasing heavy slabs in brass rolling, coils tend to become top-yeavy and unstable, and the future trend will probably be toward undercoiling, as in steel practice.

The gage at which metal is blocked varies at the different plants, but ordinarily blocking begins at around 0.060 inch in thickness. As the leading end of the strip emerges from the rolls it is caught by a "clamshell", or belt-type wrapper, which automatically wraps the first two or three layers onto the block, after which the tension of the block keeps the strip tightly wound thereon. By automatically wrapping in this manner it is possible to enter a strip without reducing the speed of the rolls.

The block on the discharge side of the rolls is driven at a speed faster than that of the rolls, thus producing a tension on the strip, which is desirable for gage control and flatness, and is preferably separately driven from its own motor, through a friction clutch. This compensates for the change in peripheral speed as the coil diameter increases and also permits manual variation of tension.

Electrically controlled tension blockers are replacing old-style blockers in order to obtain finer control of tension.

Roll coolants: When rolling metal, it is customary in most applications to apply oil to the metal as it enters the rolls. The viscosity of the oil used varies, depending on the class of work to be rolled and the reduction required per pass. For heavy rundown work, a viscosity of about 180 seconds, and for light finishing passes a viscosity of 60 to 100 seconds, is normally used; this latter can ordinarily be obtained by "cutting" the heavier oil with a light oil such as kerosene. Soluble oils - that is, animal or vegetable oils saponified to make them miscible with water - are also used in finishing operations.

The function of roll oil is twofold: (1) It assists in cooling the roll surface and stock; and (2) it acts as a cushion between the rolls and metal to assist in producing uniform pressure over the surfaces in contact with the rolls.

An indirect function of roll oils is to keep the metal out of contact with the roll surface and thus prevent "brassing" of the rolls. In the absence of oil, superficial adhesion takes place, and after a certain quantity of metal has passed through the rolls brass or copper is deposited on the roll surface, which, if not removed, causes variations in gage and other undesirable effects.

Metal rolled with oil ordinarily will have a duller surface, whereas when rolled dry or with water, or with an extremely light oil, the metal will take on the characteristics of the roll surface and if this is highly polished will result in a similarly polished

appearance on the surface of the metal. Any minor scratches or other imperfections on the roll are more likely to be reproduced on the metal surface than would be the case if an oil film were interposed.

Pickling equipment: Pickling is done in several types of machines. For heavy-gage metal the continuous-type machine is employed. The metal is fed into a roll leveler through troughs containing the pickle solution, which is recirculated, then through troughs containing water (cold and hot), to wash off the acid remaining on the surface, then through brushes and driers; it is coiled as it leaves the machine.

For metal 0.030 inch and thinner, the pull-through-type machine is employed. The starting end of the coil of metal is attached to the end of the previous coil and is pulled through the pickling solution, cold and hot water wash, brushes, and drier and wound up into coils on spindles at the delivery end of the machine.

Very little metal is pickled on racks in tubs, owing principally to the weight of coils now being handled and the fact that this method is very inefficient.

In general, pickle solutions are mixtures of sulfuric acid, nitric acid, potassium bichromate, and water.

Copper and copper alloys are generally pickled in 5- to 10-percent sulfuric acid at room temperature up to 125° to 150° F. Various other pickling solutions are also in use.

Finishing equipment: Finishing equipment may consist of slitters, shears, roller levelers, stretcher levelers, and saws.

Slitting is done to bring rolled metal to finished width. The machines used for slitting consist of rotary cutting shears mounted on two parallel shafts, which are rotated in opposite directions and driven by a motor through suitable reduction gearing. The rotary cutters are from about 2 to 16 inches in diameter, and from 1/8 to 3/4 inch thick. A coiler or winder mechanism at the exit end of the slitter is used for rewinding the slit width.

Shears are used for cutting sheet metal to finished size and for flat strips. Metal to be sheared is generally flattened before cutting to size by passing it through a roll leveler consisting of five to nine or more rolls. Roll levelers are also used to straighten and flatten strip material.

Sheets are seldom rolled flat enough to meet customer requirements and are either leveled by passing them through a roll leveler

consisting of 9 to 17 or more rolls or by patent leveling. Patent leveling is done by gripping each end of the sheet with a pair of jaws and pulling or stretching the metal until it is flat. This operation cannot be performed on metals that are very hard. The degree of hardness that can be handled depends upon the mixture and gage. Roll levelers are sometimes used to remove camber from strip metal that is sheared to specific length.

Plates are generally sawed to final size by hand or circular saws. Strip and sheets too heavy for slitting or shearing are also sawed to final width and length.

Examples of rolling practice: Rolling-mill practices are determined by the form and composition of the product and the final gage, temper, surface requirement, and quality, as well as upon the type of equipment available.

Breakdown practices are closely similar for all types of rolling, with variations as to degree of reduction, whether hot or cold rolling, and other factors.

Strip metal is cold-rolled after breakdown to finish. The number of roll passes, anneals, and pickling operations is determined by the methods employed, the machinery and equipment utilized, and, to some extent, by the layout.

Sheets of copper or brass may be hot- or cold-rolled, after breakdown to finish in single sheets or in packs of two or more. Soft sheet copper may be cold-rolled to finish, with one intermediate annealing and one finish anneal. If cold-rolled temper is required, it is obtained by one light pass on a dry roll. When accuracy of final gage for sheet copper is not essential, as average weight per square foot, the practice of hot rolling to finish is sometimes followed.

Sheet brass, when hot-rolled after breakdown, is usually taken only to the ready-to-finish gage and then cold-rolled to finish in single sheets on dry rolls. Owing to the difficulty of milling wide brass sheets after the breakdown operation, it is the usual practice to "scalp" the surface of the casting before rolling when the surface area is smaller.

The rolling of alloy plates is the same in principle as sheet rolling; that is, the castings are scalped before rolling, brought to size by cross rolling and finished either hot or cold, depending on the physical requirement. After rolling to gage, they are flattened through heavy roll levelers. Plates for such items as condenser heads are frequently cut to specified shape by a band saw or on a boring mill as a final operation.

6. Wire Fabrication

Wire is fabricated by hot-rolling or by hot extrusion of billets into rods about $\frac{1}{4}$ inch in diameter, which are then cold-drawn through various stages into wire. Extrusion is usually employed when alloy rods are produced, and in the same manner as described for the production of rods for screw machines and forgings.

After the wire is drawn to finish size, it may receive one or more of the following operations before shipping: Enameling, tinning, stranding, insulated with rubber, paper, plastic, glass fiber, silk, or cotton covering, These operations are not described herein.

A composite flow sheet of wire fabrication is shown in figure 55.

The major steps in wire production are as follows: Preliminary forming by rolling, extrusion; cold drawing, including pointing, pickling, and annealing; finishing by enameling, tinning, or insulation.

Preliminary Forming by Rolling

The wire bars used for rolling into rods for further fabrication into wire weigh about 250 pounds each as received from the refineries. After heating to a predetermined temperature the bars are delivered to a breakdown mill.

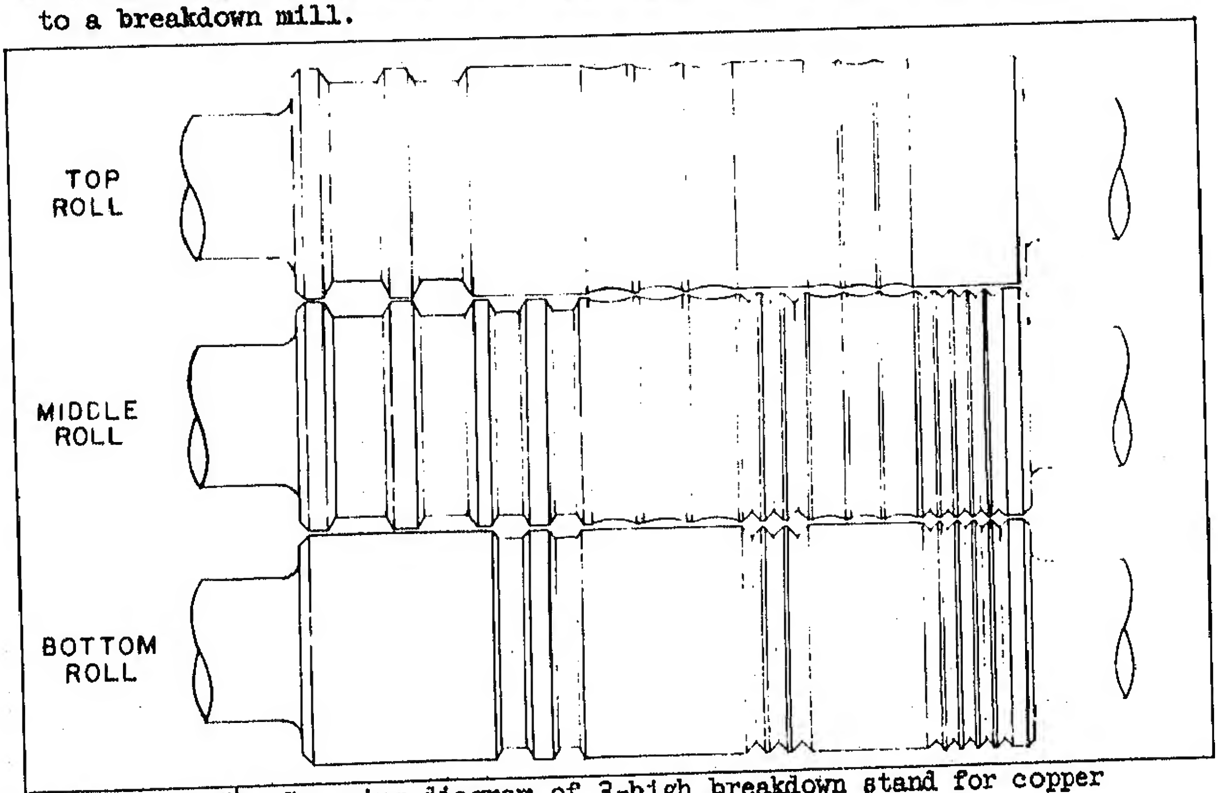


Figure II-54 Grooving diagram of 3-high breakdown stand for copper rod mill

FIGURE II-55
WIRE FABRICATION

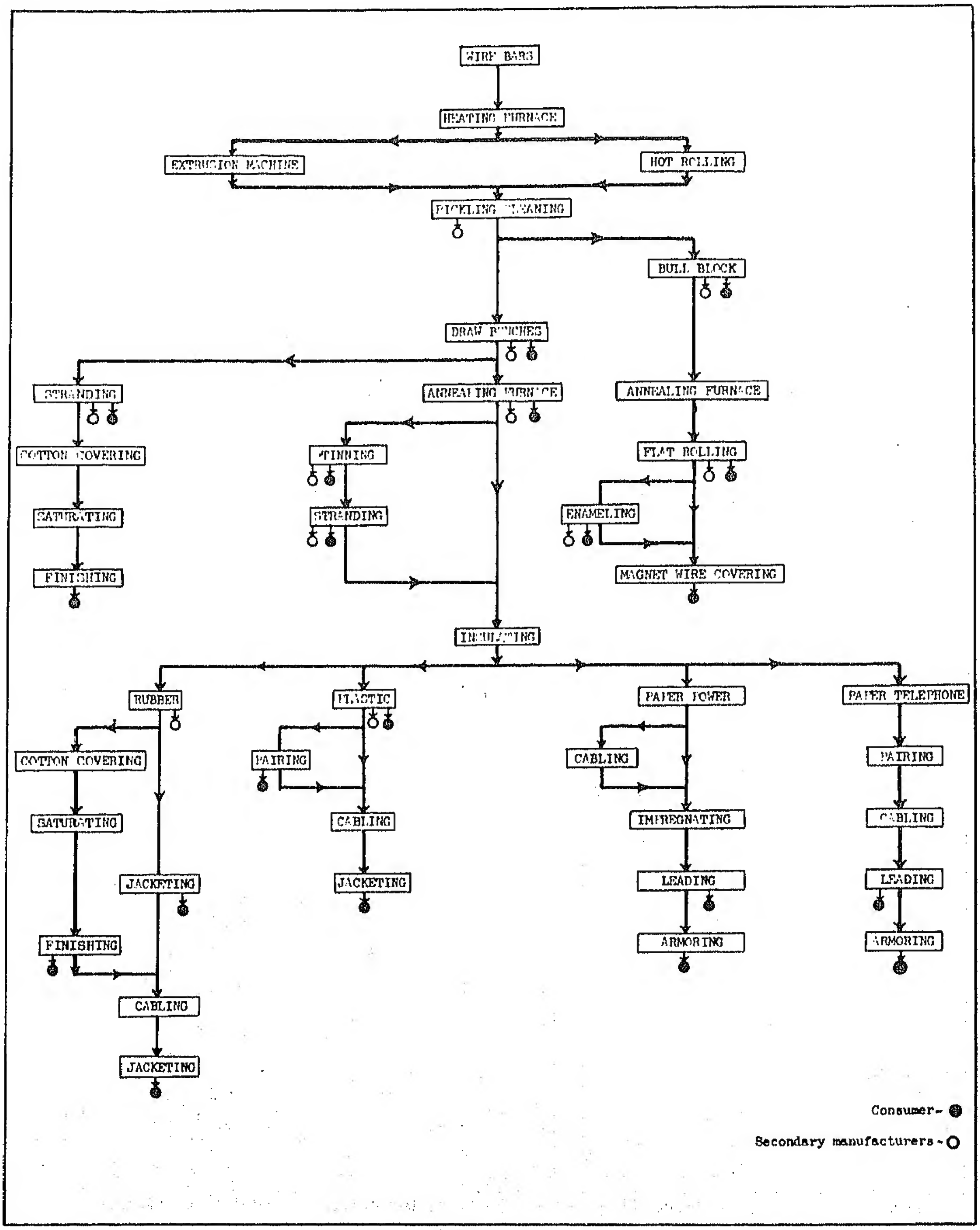


Figure II-55 Flow sheet of wire fabrication

Figure 54 is a grooving diagram for an 18-inch 3-high copper-rod breakdown mill, showing the rolls and the grooving which is required for reducing a standard wire bar having a cross sectional area of approximately 16 square inches to 1 square inch. Ordinarily, two rod strands are run simultaneously in this mill.

Rod coilers: Two different types of coilers are ordinarily used for the coiling of rod. One is the usual platen raising type, which consists of a unit on which the coils are wound on top of a platen, after which the platen is raised to a definite position and the coil stripped off by means of an air-operated stripper onto a conveying unit, which takes it through a cooling bath and out to the final tie-up position. There is some objection to the use of this type of coiler, inasmuch as it requires the use of a pusher to push the coil from the coiler to the conveyor, which may cause scratching of the rod.

The other type of coiler is the bottom drop coiler, where the coil after being completely wound, is dropped directly into the cooling bath and onto a conveyor. This coiler has fingers which open downward and outward, allowing the coil to be dropped. This type of coiler eliminates movement of the rod across the platens and onto conveyors and also has the added advantage of being able to project the coil directly into the cooling bath and on top of a conveyor that passes through the cooling bath, from where it is raised up to the floor level, tied and removed.

After cooling, the rods are then put through a series of cold-drawing operations to finish wire size.

Drawing

Various types of machines and equipment are utilized in processing the hot-rolled or extruded rod to finished wire sizes, such as bull blocks, tandem rolling machines, wire-drawing benches, and continuous drawing machines operated in tandem or single with multiple-die arrangements and combinations. They are generally of standard design. In a few cases specially designed machinery is employed for specific purposes.

Bull blocks: Bull blocks are normally used for making single reductions on rods after the hot-rolling or extrusion. They are also extensively used in the manufacturing of trolley wire, in which case it is usual to place two or more machines in tandem with the block running at different speeds to take up the elongation. The finished product is then led onto a winder drum. Figure 56 illustrates a vertical bull block without draw-out motion.

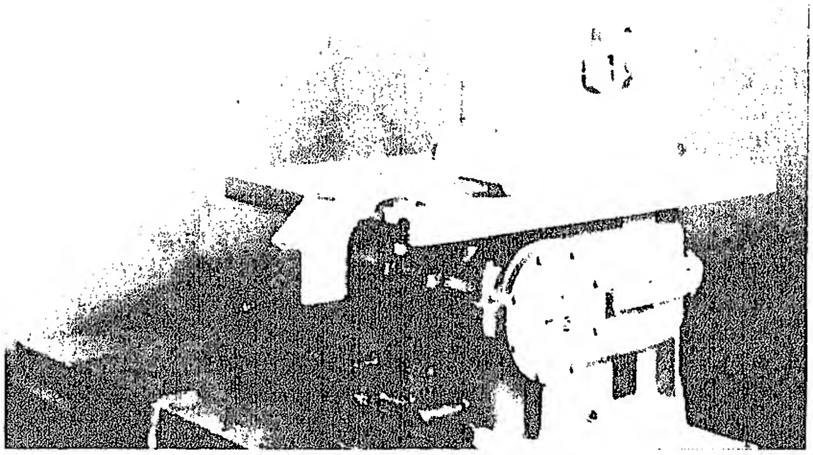


Figure II-56 Vertical bull block

Tandem wire-rolling mills: These machines are used in some instances instead of bull blocks, especially on alloys. The rod is given a series of passes through rolls set in housings, and each successive housing is set at right angles to the adjacent housing or housings. By this method, the rod can be rolled on one side and then on the other 90° apart, permitting the rod to pass through the mill in a straight line. The rod is reduced approximately 15 percent in area by each pair of rolls and requires no pointing. The cost of reducing wire by rolling in this type of machine is usually less than drawing on a bull block. These mills are built in several sizes, ranging from 6 to 12 stands. Figure 57 illustrates a 10-stand mill equipped with a straightening and feeding device and coiler.

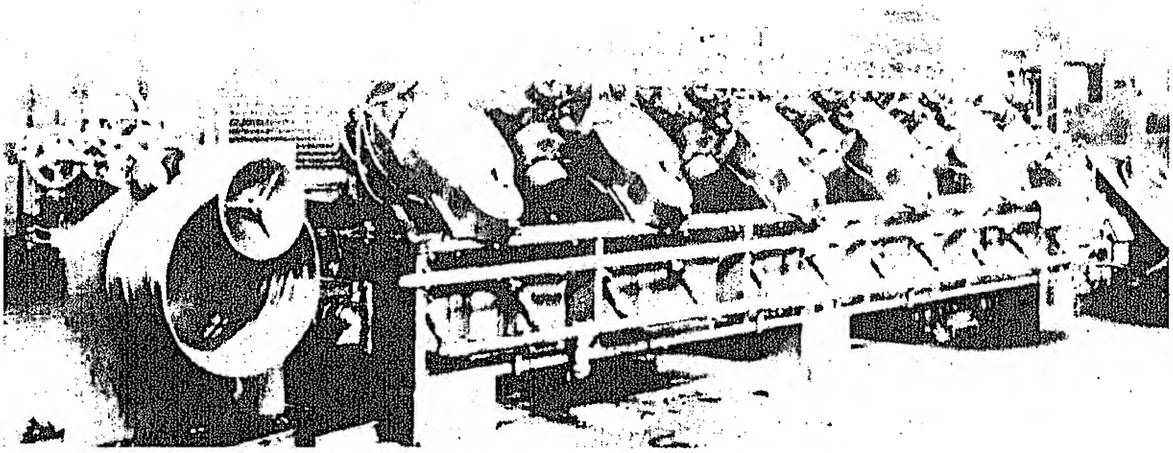


Figure II-57 Tandem wire rolling machine

Wire-rod rolling mills are designed to effect thorough hot working of the metal coincident with size reduction. In the breakdown mills, the initially square section of the wire bars is progressively changed to an elongated hexagon, an oval, a square, an oval, and finally to a square. In the rundown and finishing mills, square and oval shapès alternate until the final round section section is formed. In addition, the rod is given a 180° twist between successive passes, which is done automatically with twisting guides.

The breakdown mills are followed by a succession of rundown and finishing mills to complete reduction of the wire bars to wire rods.

The mills may be arranged in a straight line or train. A typical train consists of one 3-high breakdown mill, one or two 2-high rundown mills, and eight or more 2-high intermediate and finishing mills, requiring fifteen to twenty men for operation.

Modern production requirements have led to development of a tandem arrangement of mills. A typical tandem assembly, as shown in figure 58, comprises one 3-high rundown and eight 2-high intermediate and finishing mills, and four coilers. This arrangement is primarily for the production of 1/4-inch round rods and requires not more than eight men. Rounds can also be coiled off at a number of intermediate sizes between 1/4 and 7/8 inch. To facilitate coiling of rods of various diameters, two of the four coilers are movable, and two are stationary.

<u>Wire-drawing benches</u>: Wire-drawing benches generally are used for making a series of drafts on wire beyond the capacity of ordinary continuous drawing machines and are especially useful for heavy work as well as for meeting the numerous demands that continually arise where only one or two drafts are required on finer sizes. Wire-drawing benches consist essentially of a series of vertical spindle blocks arranged in a continuous frame with a single main drive. In the general operation of a wire-drawing bench with draw-out motion, the wire (which has first been pointed), is inserted through the die, gripped from the opposite side by tongs, pulled through the die, and the end fastened to the block. The block is then set in rotation by a foot treadle, and the finished product is delivered in coil form. The draw-out motion is a device for pulling a short length of wire through the die and up to the die block. For convenience of the operator, the draw-out motion should not run over 20 to 30 strokes per minute, depending on the size of the bench.

FIGURE II-58
ROD MILL

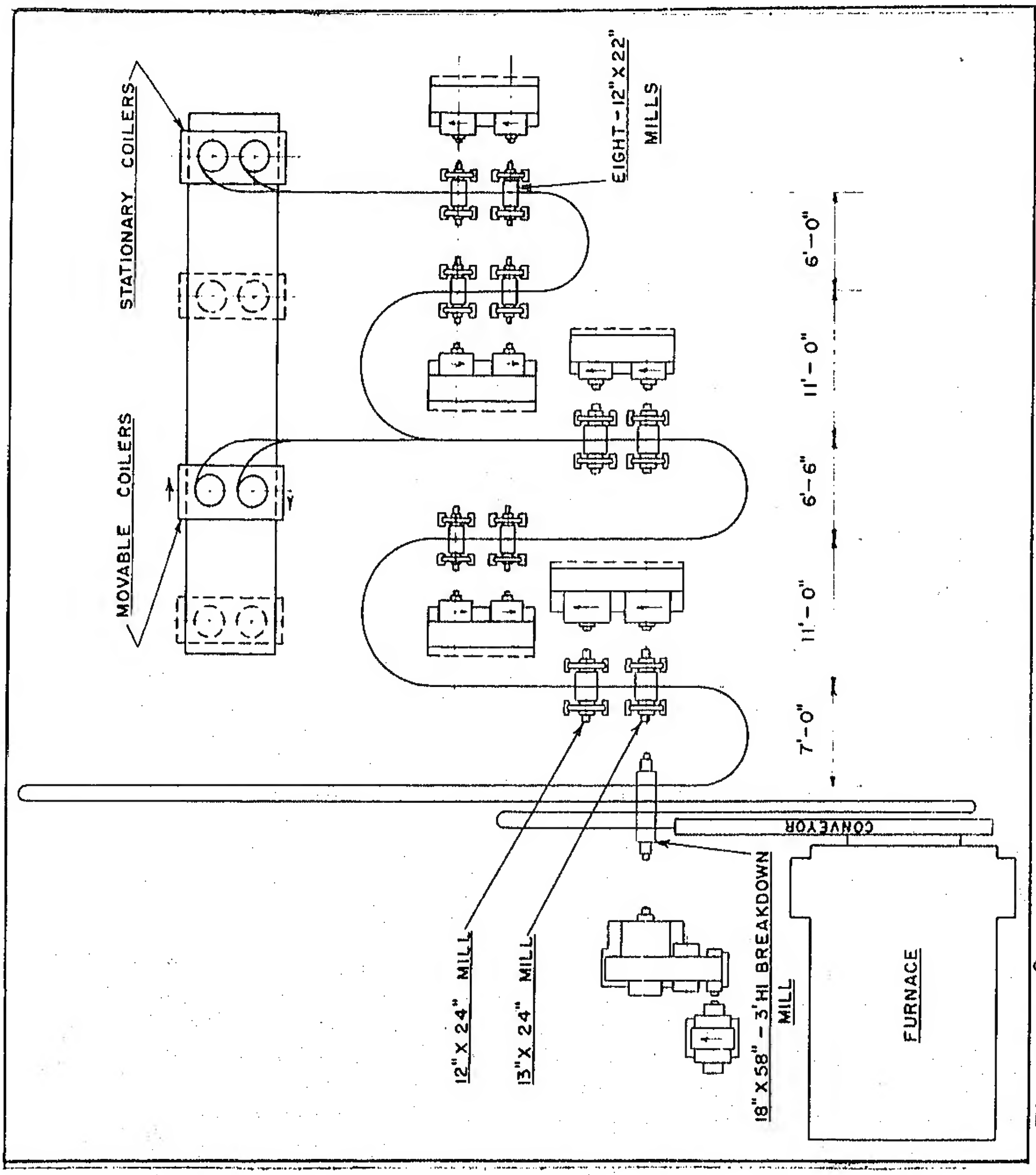


Figure II-58 Tandem wire-rod rolling mill

In the operation of a bench without draw-out motion, the end of the wire rod after being inserted through the die is gripped by a hand-wedge grip attached to the block while at rest. This arrangement, which is semiautomatic in operation, dispenses with any independent draw-out motion by tongs or other means.

Trolley-wire benches: Trolley-wire benches, as the name implies, are especially built for drawing commercial trolley wire in long lengths, using two, three, or four steel or chilled-iron dies to obtain the finished size and to meet the other requirements for standard trolley wire. The coils are brazed or welded together to meet the length or weight requirements, the passage through the die removing all evidence of the joints. In reality, a trolley-wire bench is a heavy, continuous, wire-drawing machine similar to benches with draw-out motion but arranged so that successive blocks run at increasing speeds to compensate for the elongation.

Trolley-wire benches are usually built in two sizes, based on the starting diameters of the rod, with two or more blocks.

In operating a trolley-wire bench, the annealed wire is started in the first die at the extreme left of the bench and a sufficient length is drawn through by tongs to reach the first block where the wire is gripped. Several turns are made around this block, after which the end is passed through the second die and around the second block, and so on, the finished wire passing from the last block to the winder.

These benches are of two types of draw-out motion, the "spindle type" and the "extension type", the choice depending on the speed of the last block, which is usually about 350 f.p.m. The auxiliary equipment used in conjunction with this machine generally consists of brazing equipment using either gas or oil fuel, or preferably electric butt welders, and independent roll pointers or swaging machines for pointing the wire.

Continuous-drawing machines: Various types of continuous-drawing machines equipped with multiple-die arrangements and combinations are used in large-scale wire drawing practice. In some cases as many as 21 dies are used.

Figure 59 illustrates a 12-die wire-drawing machine that produces No. 6 to No. 16 B & S gage wire from 5/16-inch copper rod. It operates at speeds of 2,000 to 4,000 f.p.m.

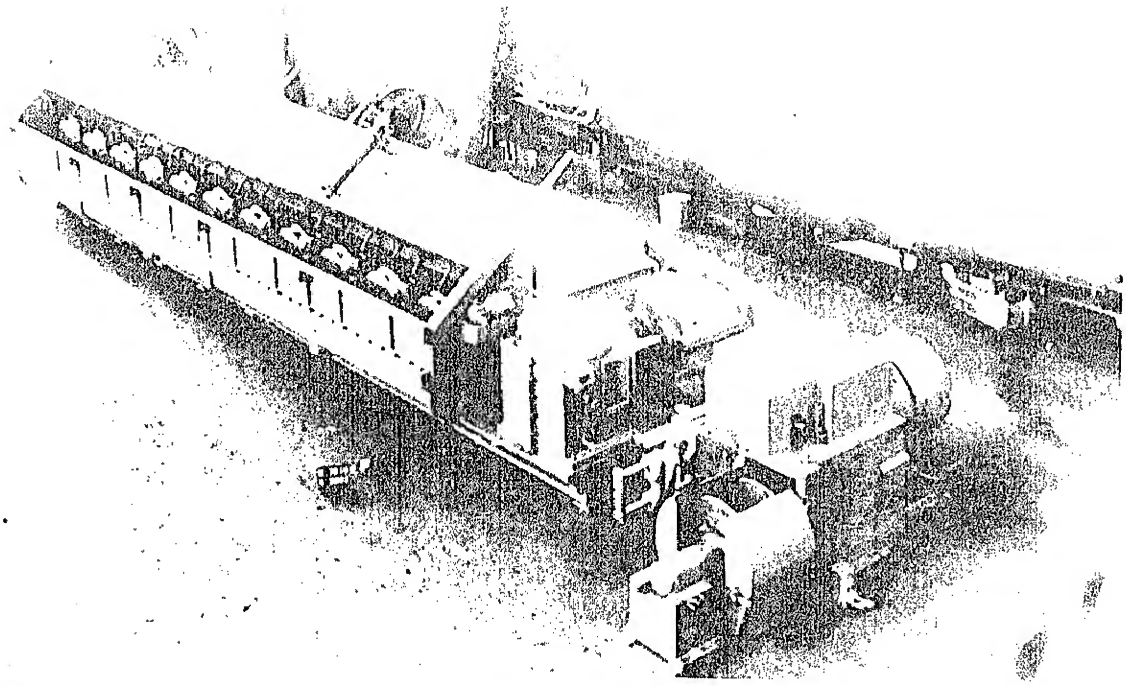


Figure II-59 12-die wire-drawing machine

Figure 60 illustrates a five-unit drawing machine with spooler.

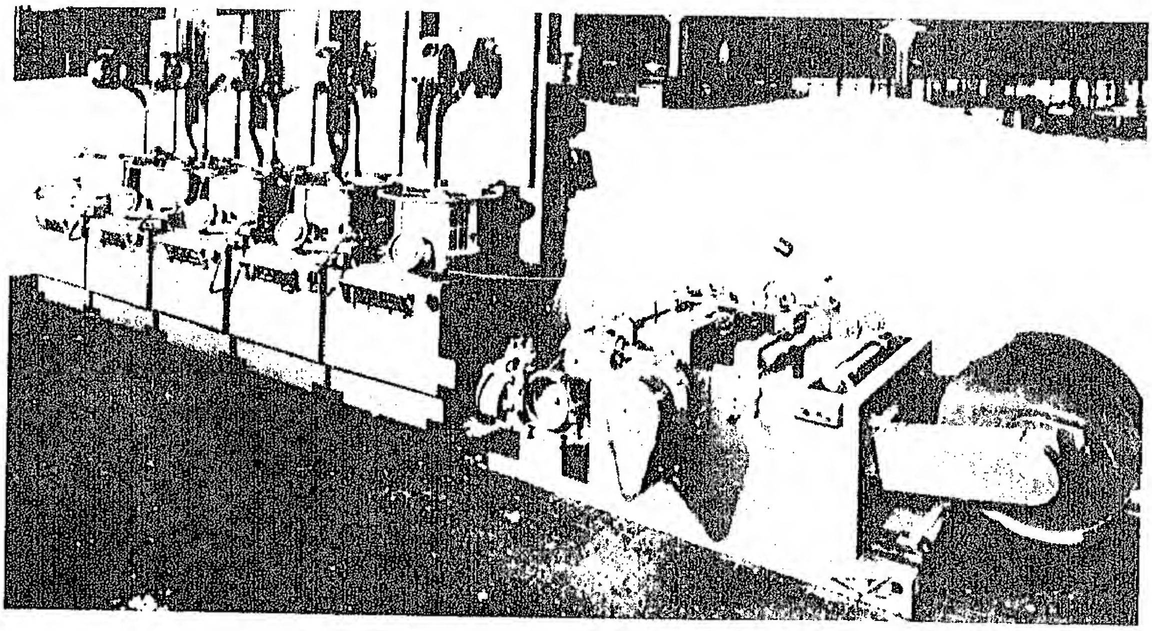


Figure II-60 5-unit wire-drawing machine with spooler

7. Bus Bars and Commutators

Bus bars, commutators, commutator segments, and other shapes and forms used by the electric industry are produced by hot rolling and cold working to finish size.

The major steps in production of these products may be classified as follows:

a. Preliminary forming by hot rolling.
b. Pickling at various stages.
c. Pointing before drawing.
d. Cold working by either
   1. rolling.
   2. drawing.
e. Annealing at various stages.
f. Finishing by
   1. Straightening.
   2. Edging.
   3. Sawing.
   4. Stamping.

The equipment and procedures are similar to those already described for the corresponding stages of rod and wire production.

8. Copper Print Rolls

Print rolls are used to print various designs on fabrics and for other printed matter. The rolls must be of a fine-grain structure, good surface, uniform properties, and free from all porosity. The method employed to produce print rolls is by hot extrusion.

Solid-cast billets are received from the refineries and vary in diameter and length. They are heated to a predetermined temperature and are extruded into a solid round to a diameter somewhat larger than the finished size. The extruded rounds are then heated and are subjected to another extrusion operation, which forms a shell with a very heavy wall.

The surface of the extruded shells is finished to size on specially designed turning lathes.

9. Scrap

In the manufacture of copper and copper-base-alloy products, approximately 50 percent scrap is generated when billets, slabs, or cakes are cast and processed into finished products. This is considered a reasonable over-all average when all products fabricated are taken into consideration.

The scrap generated for each product will vary between 30 and 60 percent, depending on the product being manufactured, the type of equipment used and the method employed. In addition, there is a zinc loss of approximately 1 percent when casting alloys. There is also a metal loss of approximately 3 percent in the course of manufacturing copper and alloys from the receipt of the raw materials to the finished products. In other words there is a total loss of metal during the processing of approximately 4 percent that is not recoverable.

The scrap generated by fabrication of copper and nonferrous alloy products is in the following form and originates from:

Melting furnaces - skimmings, spills, and drosses.
Casting - gates, physical defects, rejects due to off mixture.
Processing tube - butts and slugs from extrusion operations, pieces cut from the ends of extruded and pierced tubes, points, saw cuttings, and rejects due to off-gage surface defects, etc.
Processing rod and shapes - butts from the extruder, rod ends and points, saw cuttings, rejects due to wrong size or dimensions, etc.
Processing strip - sheets - plates - milling or scalping scrap from the overhauling of the slabs or cakes after break-down, points and tails cut from the ends of the coils or sheets before entering the rolls, trimming scrap from the slitters or shears, rejects due to off-gage, wrong temper, grain structure or other defects.

All of the above scrap is classified as plant or production scrap and is usually consumed in the plant of generation without any treatment, except skimmings, spills and drosses.

By its very nature, scrap is usually bulky and difficult to handle, and it is the general practice to cabbage or bale the light scrap and cut the heavy scrap to suitable size, so it can be handled more easily in storage and when charging.

The secondary manufacturer generates 30 to 60 percent scrap in the form of clippings, trimmings, stampings, borings, and turnings when processing copper and copper-base alloy products into semi-finished and finished articles. The actual percentage of scrap generated varies with the article being manufactured. Scrap also originates from surplus, obsolete, damaged, or idle inventory.

This type of scrap is classified as processed or new scrap and can be utilized by copper and copper-base alloy fabricators when

casting billets and slabs without any treatment except that borings and turnings are put over a magnetic separator to remove any ferrous scrap such as iron fines, fuzz or filings. There is also available at times certain types and grades of usable old scrap. All grades and types of old copper and copper-base alloy scrap that cannot be utilized by the fabricators, due to its composition, corroded condition, or if the scrap is soldered, enameled or plated, is classified as old scrap. Old or new tube scrap generally is not accepted by the fabricator unless special arrangements are made. Tinned scrap is accepted only if the tin content can be determined. Typical types of old scrap that are acceptable are fired cartridge cases, copper roofing, discarded trolley wire, transmission lines, bus bars and other utility and industrial scrap, when properly graded.

Some fabricators have installed refinery furnaces to melt down old scrap not usable due to its composition or condition, to reclaim the copper and zinc contents in order to obtain their metal requirements and to reduce their metal costs.

It is not always possible for a fabricator to obtain scrap from the various secondary manufacturers; (1) They do not keep the various grades separated and (2) in most cases they do not have enough storage space to accumulate a sizable amount for shipment back to the fabricator.

Therefore, large amounts of this new scrap are lost to the fabricators as they have no means of collecting small individual lots of scrap generated by the secondary manufacturers and other users of copper and alloy products, such as plumbers, sheet-metal concerns, and others, who in many cases obtain their materials from distributors or jobbers. The result is that this new scrap is sold to and collected by scrap dealers and used to some extent for other purposes than the manufacturing of billets, slabs, and cakes.

Apparently there is available for use by the fabricators of copper and copper-base alloy products, enough new scrap (50 percent by the fabricators and sufficient amounts generated by secondary manufacturers and others, plus usable old scrap), to permit the mill to use large quantities of scrap in place of new metals.

Specifications

The following specifications are used by some fabricators to cover scrap purchases.

General: All brass and copper scrap must be free from excess grease, oil, and other impurities. Plated, enameled, or soldered materials cannot be accepted. All scrap must be of uniform mixture

and various alloys strictly segregated. Heavy scrap, rod ends, turnings, etc., must be packed separately. Under normal conditions tube scrap is not acceptable.

In addition to the above, the following requirements must also be met:

1. Copper scrap shall be of 99.9 percent purity and consist of skeleton scrap from new sheets or strip stock.

2. Brass scrap shall consist of skeleton, trimmings, clippings, and punchings scrap from new sheet or strip. Punchings may not be smaller than $\frac{1}{4}$ inch in diameter and not to be more than 10 percent of the total shipment.

3. Turnings and borings from free-cutting brass rod shall consist solely of free-cutting turnings, free from iron, steel, aluminum, manganese and all other alloys. They shall be free of grindings and babbitts and shall contain not more than 0.30 percent tin, not more than 0.15 percent combined iron, and not more than 3 percent oil and moisture.

4. Brass forging-rod flashings shall contain not more than 10 percent punchings and not smaller than $\frac{1}{4}$ inch in diameter.

5. Brass forging-rod turnings or borings shall consist solely of rod turnings free from aluminum, manganese, etc., and all other alloys. They shall be free of grindings and babbitts and shall contain not more than 0.3 percent tin, not more than 0.15 percent combined iron, and not more than 3 percent oil and moisture.

6. Commercial bronze and low brass shall meet the same requirements as for sheet-brass scrap and must contain no tin.

Scrap should be shipped "loose", not in compressed form. Receiving weights are to govern.

Sources of Scrap

Details of the sources of fabricating-plant scrap in average practice are given in the following table (also on p. 138). The actual percentages of scrap generated depend on the equipment and practices at each plant.

AVERAGE PERCENTAGE OF SCRAP GENERATED WHEN CASTING
COPPER AND COPPER-BASE ALLOYS INTO BILLETS-SLABS-CAKES
TO BE FABRICATED INTO COPPER-ALLOY PRODUCTS

		Percent
Billets--extrusion--rod and shapes	Gates (1st cut)	3-5
All alloys	Physical defects (2d cut)	2-4
Electric furnaces	Off mixtures	½-2
Water-cooled molds	General average - all scrap	5
Air-cooled molds	Do.	8
Billets -- rolled rods	Gates (1st cut)	6-8
All alloys	Physical defects (2d cut)	3-4
Electric furnaces	Off mixtures	½-2
Air-cooled molds	General average - all scrap	10
Billets -- rolled rods	Gates (1st cut)	8-12
All alloys	Physical defects (2d cut)	4-6
Reverberatory furnaces	Off mixtures	½-2
Air-cooled molds	General average - all scrap	12
Billets--extrusion--tube	Gates (1st cut)	3-5
All alloys	Physical defects (2d cut)	1-2
Electric furnaces	Off mixtures	½-2
Water-cooled molds	General average - all scrap	6
Air-cooled molds	Do.	9
Billets--piercing--tubes	Gates (1st cut)	4-6
Copper	Physical defects (2d cut)	2-4
Electric furnaces	Off mixtures	½-2
Water-cooled molds	General average - all scrap	8
Air-cooled molds	Do.	10
Billets--piercing--tubes	Gates (1st cut)	4-6
All alloys	Physical defects (2d cut)	2-3
Electric furnaces	Off mixtures	½-2
Water-cooled molds	General average - all scrap	10
Air-cooled molds	Do.	12
Slabs--flat--for strip	Gates (1st cut)	3-5
All alloys other than rich mixtures	Physical defects (2d cut)	1-2
Electric furnaces	Off mixtures	½-1
Water-cooled molds	General average - all scrap	7
Air-cooled molds	Do.	9
Slabs--flat--for strip	Gates (1st cut)	3-5
All alloys--rich mixtures incl.	Physical defects (2d cut)	2-4
Nickel silver--electric furnaces	Off mixtures	½-2
Water-cooled molds	General average - all scrap	9
Air-cooled molds	Do.	12
Cakes--flat--for sheets and plates	Gates (1st cut)	12-18
All alloys	Physical defects (2d cut)	2-4
Air-cooled molds	Off mixtures	3-6
Electric and reveratory furnace	General average	16

PRACTICAL MAXIMUM LIMITS OF SCRAP
UTILIZATION IN COPPER AND BRASS FABRICATION

	Type of scrap	Percent
Billets for rod and shapes containing 2.25 percent lead or more	Borings Reclaims Copper or brass	50-90 10-15 5-10
Billets for rod and shapes containing 1 to 2 percent lead	Borings Reclaims Copper or brass	20-40 5-10 80-90
Billets for rod containing less than 1 percent lead	Borings Reclaims Copper or brass	10-20 - 5 70-80
Billets for rod, nonleaded	Brass Copper	30-50 30-50
Billets for tube-piercing all alloys	Brass Copper	30-50 30-60
Billets for tube-piercing copper	Copper	90
Billets for tube-extrusion copper	Copper	90
Billets for tube-extrusion all alloys	Brass Copper	30-50 30-70
Slabs--cakes for rolling brass strip and sheets 75 percent copper and under	Brass Copper	25-75 20-30
Slabs--cakes for rolling brass strip and sheets 76 percent copper and over	Brass Copper	20-50 25-60
Slabs--cakes--hot-rolled--strip and sheets, all mixtures	Brass Copper	25-50 25-50
Slabs--cakes for rolling lead brass strip and sheets	Brass Copper	25-75 20-40
Slabs--cakes for rolling nickel-silver strip and sheets	Brass Copper Nickel	20-30 50-80 10-15
Slabs--cakes--muntz metal strip, sheets, and plates	Brass Copper	40-50 50-60

Limits of Scrap Utilization

Fabricators prefer to utilize as much clean selected scrap as possible to conserve new metal, which is more expensive than scrap. Certain technical limitations apply to the maximum copper and alloy-scrap ratios to new metal that can be utilized, but it is generally possible to use up to 90 or 100 percent scrap if it is available in usable form and composition.

It is generally assumed that at least 1 cent per pound can be saved in metal cost when scrap is used in place of virgin metal. However, casting shops are reluctant to use the maximum amount of scrap permissible, as it requires a little more manual effort to charge the furnace. Also the melting time may be slightly longer if the greater percentage of the scrap is fine or loose, resulting in fewer pounds produced per furnace and man-hour; however, with proper control, production and quality are not affected.

The practical limits of scrap utilization for the fabricators of various copper and brass products is given in the following table.

AVERAGE PERCENTAGES OF SCRAP GENERATED IN FABRICATING COPPER AND ALLOY PRODUCTS FROM THE CAST BILLET OR SLAB TO FINISHED SIZE

	Percent
Average percentage of scrap generated when producing strip 20 inches wide and narrower, all gages, all alloys	30-45
Average percentage of scrap generated when producing sheet, all gages, all alloys, wider than 20 inches	30-50
Average percentage of scrap generated when producing copper strip 20 inches wide and narrower, all gages	25-40
Average percentage of scrap generated when producing copper sheets wider than 20 inches	30-45
Average percentage of scrap generated when producing alloy tubes, all sizes and gages	30-55
Average percentage of scrap generated when producing plates, all mixtures, all sizes and shapes	40-60
Average percentage of scrap generated when producing copper tubes, all sizes and gages	25-40
Average percentage of scrap generated when producing rods, all alloys, all sizes	25-35
Average percentage of scrap generated when producing shapes, all alloys, all sizes	30-50

All of the scrap in the preceding table is considered production or plant scrap and is utilized by the plant when melting new charges. In addition, there is generated in melting approximately 2 to 4 percent dross and skimmings. About one-third of the weight of dross and skimmings is zinc oxide and ash from charcoal, and almost all of the rest is small particles of metal. The metal content of the skimmings averages between 30 and 40 percent, about half or three-fourths of which is usually reclaimed by passing it through a ball mill and shakers and over screens. The remainder of the material is shipped to the refineries, where the copper content is recovered and usually returned to the original shipper.

10. Floor Plans of Typical Fabricating Plants

Typical floor plans of fabricating plants for the production of copper and copper-base alloy products are shown in figures 61 to 65.

Figure 61 shows a floor plan of a mill for producing seamless tubing, the dotted lines indicating the flow of material from the receiving of refined metals and scrap to the shipment of the finished products. The casting shop supplies the mill with rough forms (round billets), to be fabricated into tubes; some copper billets are furnished by the refineries.

The estimated monthly capacity, based on a standard pattern of orders and with an efficient lay-out of machinery and equipment, is 2,500,000 pounds.

Approximate number of workers per shift, not including supervision

Receiving, storage, and cast shop	75
Tube mill	600
Total number of workers	675

Approximate connected horsepower

Cast shop	4,000
Tube mill	6,500
Total connected load	10,500

Estimated cost of machinery and equipment would exceed $5,000,000 on the basis of 1950 costs.

Approximate cost of a modern steel and brick building designed for cranes based on the total square feet shown in drawing (215,000 sq. ft.), at a cost of $12 per square foot - $2,580,000.

FIGURE II-61
TUBE MILL

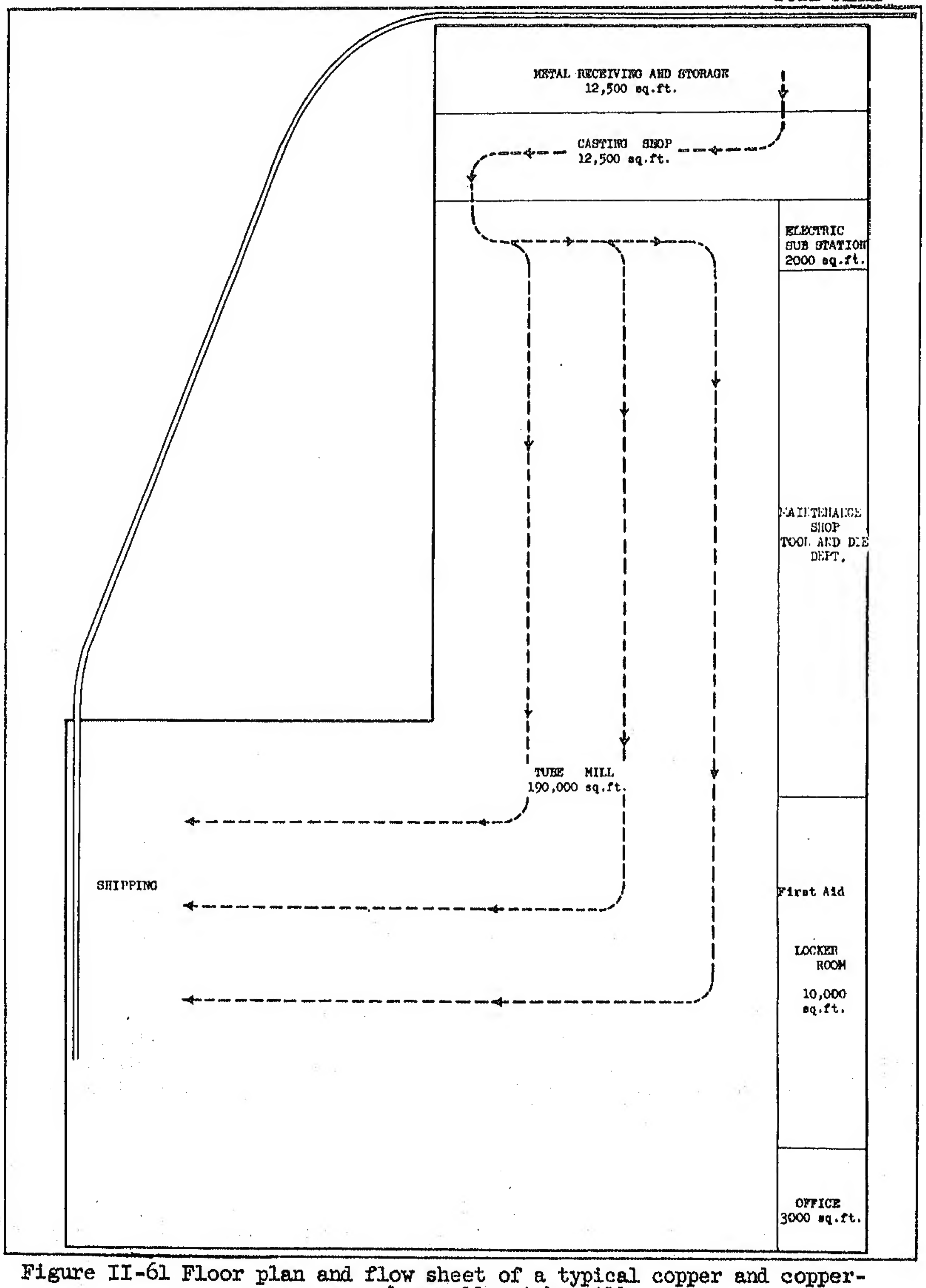


Figure II-61 Floor plan and flow sheet of a typical copper and copper-base-alloy tube mill

Figure 62 shows a floor plan of a mill for producing extruded rods, shapes, and for rolling strip materials, the dotted lines indicating the flow of materials from the receiving of refined metals and scrap to the shipping of the finished product. The casting shop supplies the mill with rough forms (round billets), for the rod and shape mill and alloy slabs for the strip mill. Copper cakes to be rolled into strip are obtained from the refineries. The machine shop maintains the machinery in good operating condition and makes the necessary repairs. The tool and die department makes all the dies for the rod and shape mill.

The estimated monthly capacity based on a standard pattern of orders is:

	Pounds
Extruded rods and shapes	3,000,000
Copper strip (maximum width, 20 inches)	1,500,000
Alloy strip (maximum width, 18 inches)	3,500,000
Total estimated capacity	8,000,000

Approximate number of workers per shift, not including supervision

Rod and shape mill	170
Rolling mill	350
Casting shop	125
Machine shop and die department	150
Total number of workers	795

Approximate connected horsepower

Rod and shape mill	2,000
Rolling mill	9,500
Casting shop	5,200
Machine shop and die department	800
Total connected load	17,500

Estimate cost of machinery and equipment would exceed $6,000,000.

Approximate cost of a modern steel and brick building based on the total square feet shown in drawing (328,750 sq. ft.), at a cost of $12 per square foot - $4,109,375.

Figure 62 shows a floor plan of a mill for the production of copper sheets and copper and alloy strip materials, the dotted lines indicating the flow of materials from the receiving of refined metals and scrap to the shipping of the finished products. The casting shop

FIGURE II-62
STRIP ROD AND SHAPE MILL

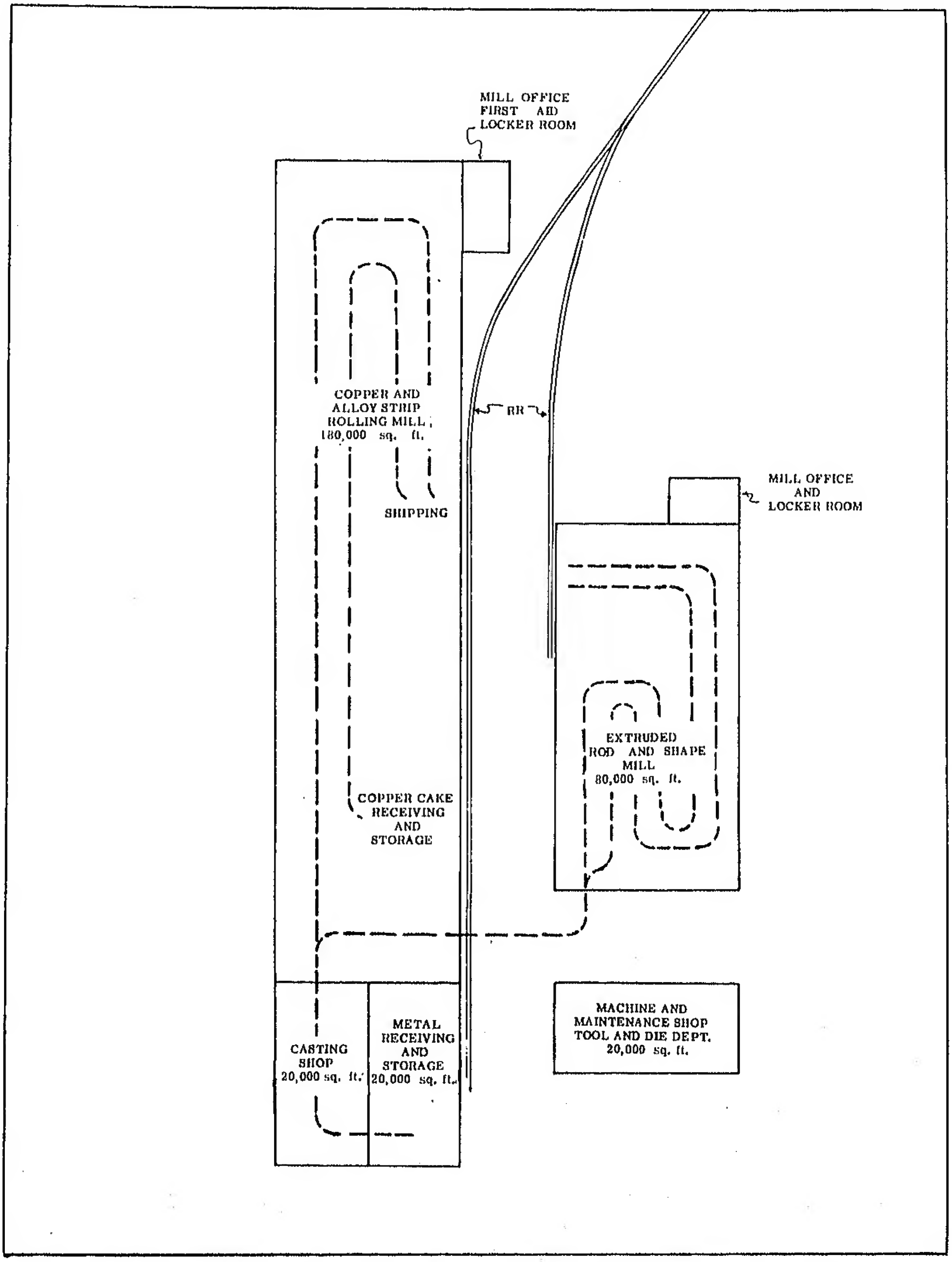


Figure II-62 Floor plan and flow sheet of a typical copper and alloy strip and extruded rod and shape mill

supplies alloy cast slabs to be rolled into strip. Copper cakes to be rolled into sheets and strip are obtained from the refineries. The machine and maintainance department keeps the equipment in good operating condition.

The estimated monthly capacity, based on a standard pattern of orders is:

	Pounds
Copper sheets (maximum width, 36 inches)	2,500,000
Copper strip (maximum width, 20 inches)	1,500,000
Alloy strip (maximum width, 20 inches	2,500,000
Total estimated capacity	6,500,000

Approximate number of workers per shift, not including supervision

Rolling mill and shipping department	425
Casting shop	75
Machine and maintainance department	100
Total number of workers	600

Approximate connected horsepower

Rolling mill	12,000
Casting shop	4,000
Machine shop	600
Total connected load	16,600

Estimate cost of machinery and equipment would exceed $7,000,000.

Approximate cost of a steel and brick building designed for cranes based on the total square feet shown in drawing (260,000 square feet), at a cost of $12 per square foot - $3,120,000.

Figure 63 shows a floor plan of a mill for the fabrication of copper and alloy strip materials, the dotted lines indicating the flow of materials from the receiving of refined metals and scrap to the shipping of the finished products. The casting shop produces the alloy slabs to be rolled into strip and copper cakes are obtained from the refineries.

The estimated monthly capacity based on a standard pattern of orders is:

	Pounds
Copper strip (maximum width, 20 inches)	2,000,000
Alloy strip (maximum width, 15 inches)	1,500,000
Total	3,500,000

FIGURE II-63
SHEET AND STRIP MILL

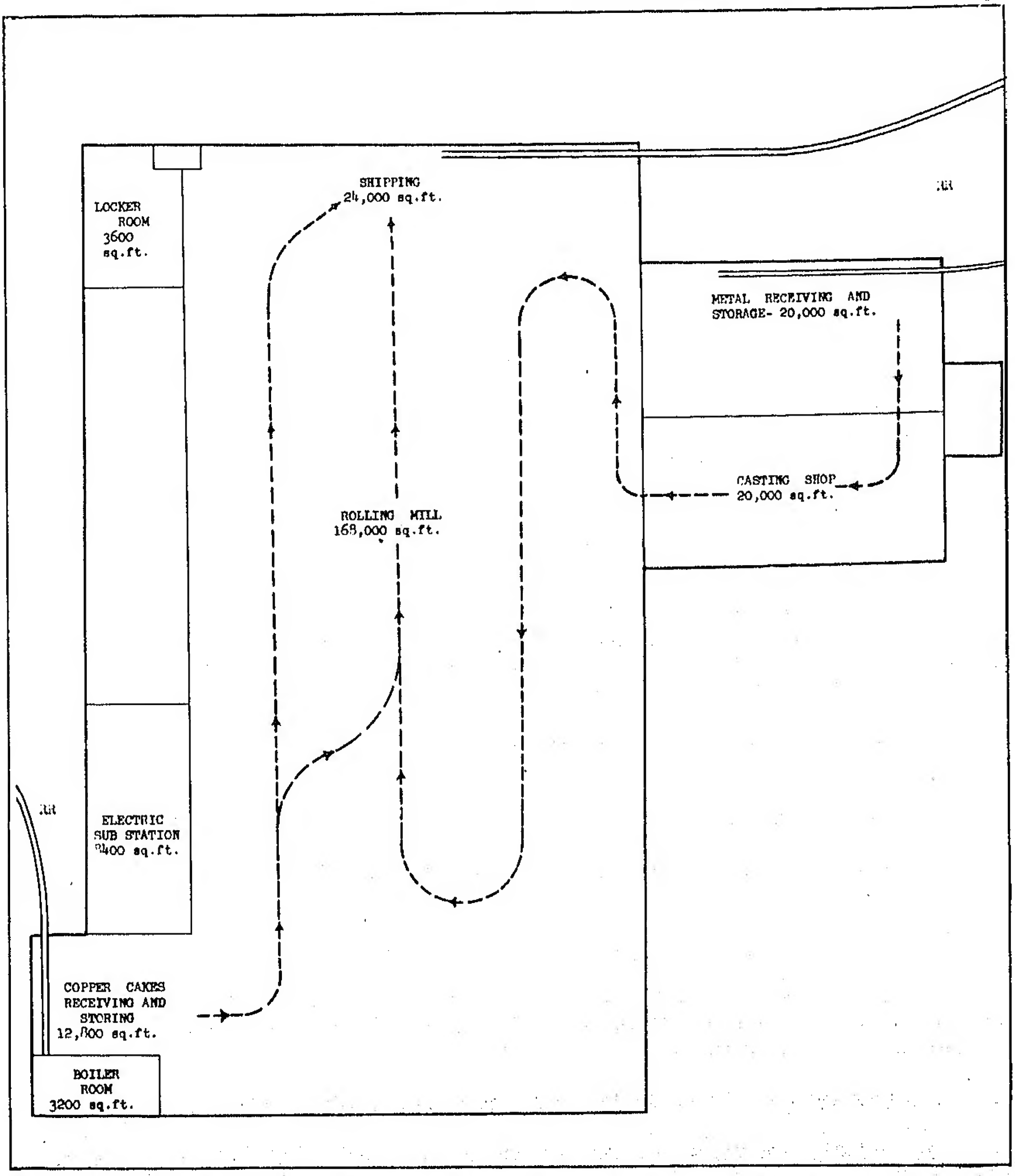


Figure II-63 Floor plan and flow sheet of a typical copper sheet, strip and alloy strip mill

Approximate number of workers per shift, not including supervision

Casting shop	75
Mill	300
Total number	375

Approximate connected horsepower

Casting shop	3,500
Mill	8,000
Total connected load	11,500

Estimate cost of machinery and equipment would exceed $4,000,000.

Approximate cost of a modern steel and brick building based on the total square feet shown in drawing (180,000 square feet) at a cost of $12 per square foot - $2,160,000.

Figure 64 shows a floor plan of a mill (Government-financed Plancor No. 91) for the fabrication of strip brass and the production of 30- and 50-caliber cups.

The brass slabs cast in this plant weighed 2,000 pounds each and were hot rolled to approximate 0.5-inch gage, at which point the surfaces were milled to remove the cast structure. The metal was then cold-rolled to finish gage.

Approximate number of workers per shift, not including supervision, 350.

The average monthly production of rolled strip, 20,000,000 pounds.

The total cost of machinery, equipment, and building exceeded $12,000,000.

or

Figure 64 shows a floor plan of a Government-financed (Plancor No. 91) mill for the fabrication of strip brass and the production of 30- and 50-caliber cups.

The average monthly production of strip brass, 20,000,000 pounds.

Approximate number of workers per shift, not including supervision, 350.

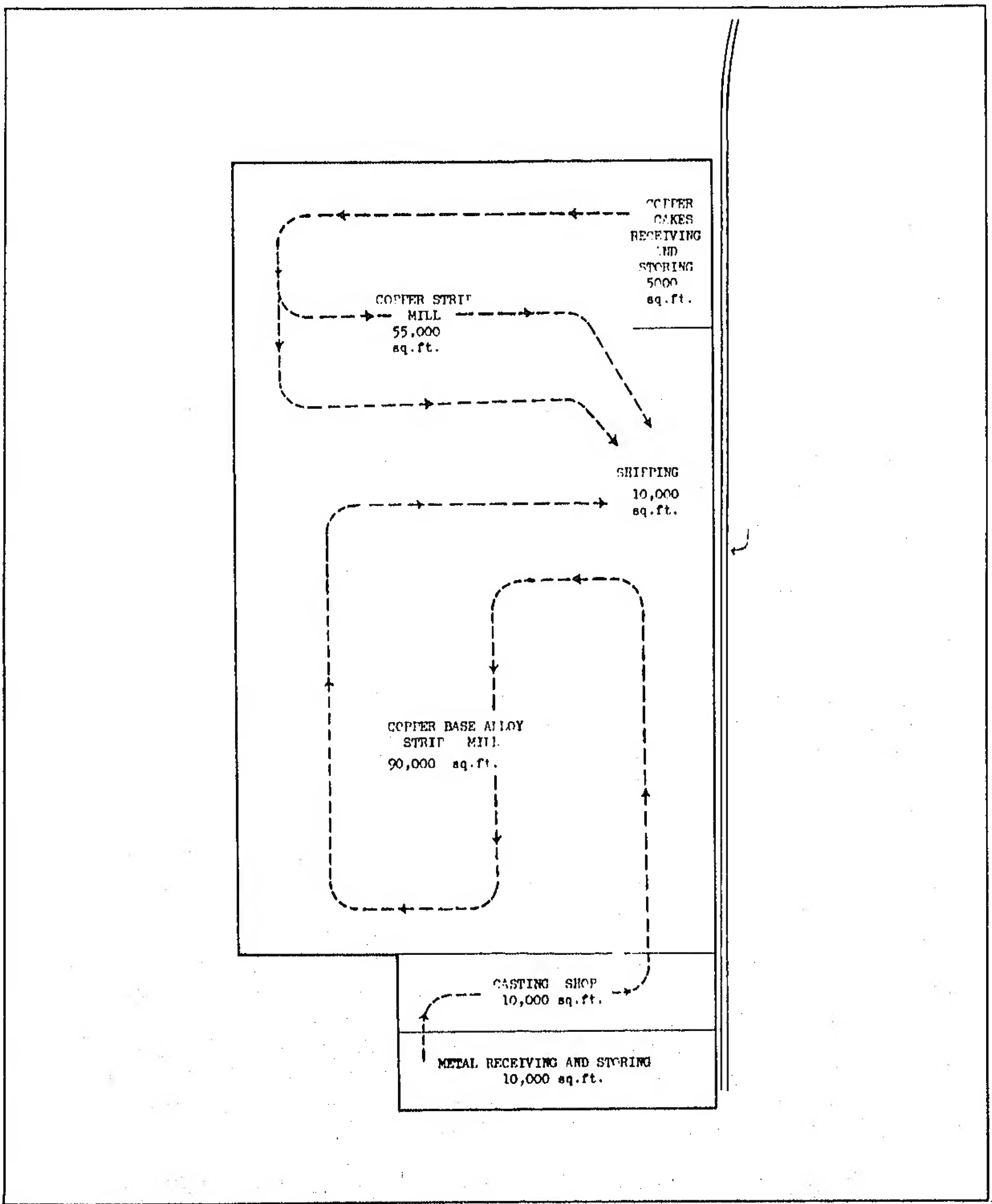


Figure II-64 Floor plan and flow sheet of a typical copper and copper-base-alloy strip mill

The total cost of machinery, equipment and building exceeded $12,000,000.

Figure 65 shows a floor plan of a mill to produce wire products and lines indicating the flow of materials through the various operations to shipment of the finished products.

11. Cost

The cost of converting copper and brass into fabricated products varies between two companies and even between two plants of the same company on account of divergences in fabricating and handling equipment, processes and the scale of production which influence the output per man and per machine. Even with closely parallel processes, reported costs may vary considerably with accounting practices, though the true costs on a comparable basis may be nearly equal.

To arrive at the total cost of a manufactured article, it is customary to group expenditures into several classifications - material, (metal), direct labor, plant overhead or burden, selling, and administration expenses.

In manufacturing copper and alloy products, the cost of the metal represents the largest single portion of the total cost of any article. For items that require a large number of operations, the cost of fabrication frequently is greater than the cost of metal.

Inasmuch as prices of the finished products reflect the cost of manufacturing, current price quotations, indicating the differentials between the sale price and the cost of raw materials, are a good guide from which to average fabricating costs, including profit margins.

The differential or spread between the selling price and metal cost generally remains constant, and there is a definite relationship, with very few exceptions, between the cost of metal and the net selling price. If and when the differential changes, it is generally due to one or combinations of the following factors: hourly wage rates, transportation costs, increase or decrease in the cost of materials (other than metal) and supplies, such as power, gas, oils, lumber, etc. In exceptional cases the differentials have changed, owing to customer requirements necessitating extra operations, technological changes in machinery or methods of production, and just recently because of a retirement plan that was put in effect.

The following tables give the December 1, 1950, price quotation of the principal sizes and types of copper and alloy products and

FIGURE II-65
WIRE MILL

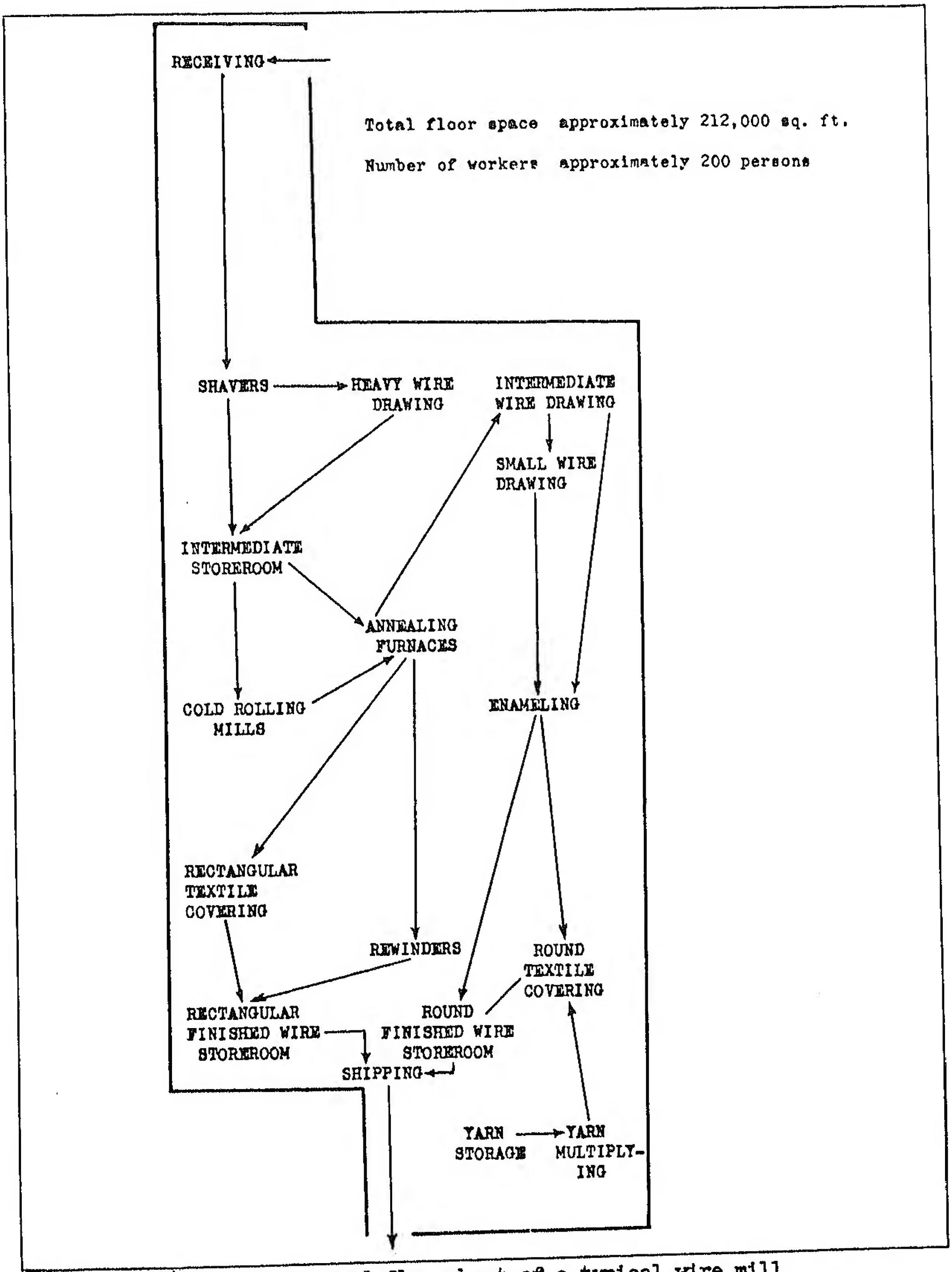


Figure II-65 Floor plan and flow sheet of a typical wire mill

the differential or spread between selling price and metal cost. These data serve as an index of fabricating costs and are useful in estimating the probable effect on costs of changes in the industry.

<u>Differential or spread between selling price and metal cost of some principal items or products manufactured by copper and brass fabricators</u>

Selling prices in effect as of--December 1, 1950.

Metal values in effect as of ---October 19, 1950.
Copper 24.50¢
Zinc 18.47¢
Nickel 48.88¢
Tin $1.12
Aluminum (for brass and bronze alloys) 19.00¢

Items listing distributor prices are sold exclusively through distributors. Other prices listed are those paid by consumers.

Quantity schedules are based on single item basis, for complete shipment at one time to one destination. On alloy rods, in round, hexagon square - single order basis applies. The term "Order" means the amount contained in one order for one alloy only, in varying sizes for shipment at one time to one destination, provided no item on the order is for less than 100 pounds.

Extras are charged for special operations on tubing and pipe such as:

Polishing, tinning, plating, threading, cutting pipe under 1 foot, also for tinned and lead-coated sheet or strip.

Copper Water Tube

Type K - for underground and interior service.
Type L - for interior service.
Type M - for soil, waste and vent service.

Distributor price - 2,000 or more feet or pounds.

Nominal size, inches	Selling price per foot	Value of metal, cents per lb.	Differential spread, cents per lb.
1 K	35.21¢	24.50	17.467
2 K	83.98¢	24.50	16.267
3 K	$1.5828	24.50	15.070
½ L	14.69¢	24.50	27.044
1 L	29.06¢	24.50	19.866
2 L	75.54¢	24.50	18.666
1¼M	30.26¢	24.50	19.870
2 M	64.78¢	24.50	19.870
3 M	$1.1568	24.50	18.668
4 M	$2.0117	24.50	18.670

Copper Tube for Oil-Burner Use

Distributor price - 2,000 or more feet or pounds.

O.D., inch	Gage, inch	Selling price per foot	Value of metal, cents per lb.	Differential or spread, cents
¼	0.049	06.81¢	24.50	32.250
3/8	0.049	10.13¢	24.50	27.449

Red Brass and Copper Pipe

Distributor price - 2,000 or more feet or pounds.

Standard pipe size, inches	Selling price per foot	Value of metal, cents per lb.	Differential or spread, cents per lb.
Red brass-reg.			
½	40.60¢	23.595	19.874
3/4	53.55¢	23.595	18.570
1¼	$1.0065	23.595	14.675
2	$1.5767	23.595	14.574
3	$3.2759	23.595	14.675
Copper-regular			
½	39.65¢	24.50	17.018
3/4	52.41¢	24.50	15.815
1¼	98.77¢	24.50	12.217
2	$1.5495	24.50	12.218
3	$3.2130	24.50	12.220
Red brass-extra strong			
½	58.26¢	23.595	23.771
3/4	70.42¢	23.595	18.573
1¼	$1.2973	23.595	14.673
2	$2.1699	23.595	14.675
3	$4.4393	23.595	14.675
Copper-extra strong			
½	54.40¢	24.50	20.620
3/4	68.94¢	24.50	15.816
1¼	$1.2705	24.50	12.220
2	$2.1297	24.50	12.219
3	$4.3329	24.50	12.219

Condenser and Heat Exchange Tubes

Quantities of 30,000 lb. or more; lengths, 1 to 30 feet, inclusive - one alloy.

Size inch O.D. - Wall		Alloy	Price per pound, cents	Value of metal, cents per lb.	Differential or spread, cents
5/8	0.049	Admiralty	50.35	23.35	26.70
3/4	0.049	Do.	49.23	23.35	25.58
7/8	0.049	Do.	49.23	23.35	25.58
1	0.049	Do.	49.23	23.35	25.58
5/8	0.049	Cupro-nickel-30%	69.24	31.76	37.48
3/4	0.049	Do.	68.12	31.76	36.36
7/8	0.049	Do.	68.12	31.76	36.36
1	0.049	Do.	68.12	31.76	36.36

Copper Refrigeration Tubes - Sealed Ends - 50-ft. Coils

Distributor price - 300 or more coils - - Order basis.

O.D. inch	Wall, inch	Price per coil	Value of metal, cents per lb.	Differential or spread, cents per lb.
1/4	0.030	$2.58	24.50	39.679
3/8	0.032	$3.82	24.50	32.515
1/2	0.032	$5.07	24.50	31.214

Soft Copper Tubes - 25- and 50-ft. Coils. For Automotive and General Use

O.D. inch	Wall, inch	Price per lb., cents	Value of metal, cents per lb.	Differential or spread, cents per lb.
1/4	0.032	60.82	24.50	36.32
3/8	0.032	56.02	24.50	31.52

Standard Stock Sizes - Sheet, Strip and Roll Copper

Distributor prices for 2,000 to 5,000 lb.; order basis, no items under 500 pounds.

Size, inches	Price per lb., cents	Value of metal, cents per lb.	Differential or spread, cents
30 X 96 X 16 oz. - soft	42.73	24.50	18.23
Do. - C.R.*	45.13	24.50	20.63
30 X 96 X 20 oz. - soft	41.78	24.50	17.28
Do. - C.R.	44.18	24.50	19.68
12 X 96 X 16 oz.	39.81	24.50	15.31
20 X 96 X 16 oz.	44.61	24.50	20.11
6,7 or 8 X 16 oz. - Rolls	37.41	24.50	12.91
18 and 20 X 16 oz.- Rolls	42.21	24.50	17.71

*C.R.=Cold-Rolled

Copper in Rolls - Miscellaneous

Consumer price - Item basis - 30,000 lb. or more.

Width, inches	Gage, inch	Price per pound, cents		Metal value, cents	Differential or spread, cents
		Rolls	Mill lengths		
Inc. ½ to 2	0.0508 & heavier	38.41	40.21	24.50	13.91 and 15.71
Inc. 2 inc. 8	0.0508 & heavier	37.21	37.81	24.50	12.71 and 13.31
Over 8 inc. 12	0.0508 & heavier	37.81	38.41	24.50	13.31 and 13.91
Over 16 inc. 20	0.0508 & heavier	39.01	39.61	24.50	14.51 and 15.11
Inc. ½ to 2	.032	38.41	40.21	24.50	13.91 and 15.71
Inc. 2 inc. 8	.032	37.81	38.41	24.50	13.31 and 13.91
Over 8 inc. 12	.032	38.41	39.01	24.50	13.91 and 14.51
Over 16 inc. 20	.032	40.21	40.81	24.50	15.71 and 16.31
Inc. ½ to 2	.0201	39.61	42.01	24.50	15.11 and 17.51
Inc. 2 inc. 8	.0201	38.41	39.01	24.50	13.91 and 14.51
Over 8 inc. 12	.0201	39.61	40.21	24.50	15.11 and 15.71
Over 16 inc. 20	.0201	43.21	43.81	24.50	18.71 and 19.31
Inc. ½ to 2	.010	41.41	45.01	24.50	16.91 and 20.51
Inc. 2 inc. 8	.010	40.21	42.01	24.50	15.71 and 17.51
Over 8 inc. 12	.010	40.81	42.61	24.50	16.31 and 18.11
Over 16 inc. 20	.010	50.41	52.21	24.50	25.91 and 27.71
Inc. 2 inc. 8	.005	43.81	48.61	24.50	19.31 and 24.11

Standard Stock Sizes - Sheet, Strip and Roll Copper

Distributor prices for 2,000 to 5,000 lb.; order basis, no items under 500 pounds.

Size, inches	Price per lb., cents	Value of metal, cents per lb.	Differential or spread, cents
30 X 96 X 16 oz. - soft	42.73	24.50	18.23
Do. - C.R.*	45.13	24.50	20.63
30 X 96 X 20 oz. - soft	41.78	24.50	17.28
Do. - C.R.	44.18	24.50	19.68
12 X 96 X 16 oz.	39.81	24.50	15.31
20 X 96 X 16 oz.	44.61	24.50	20.11
6,7 or 8 X 16 oz. - Rolls	37.41	24.50	12.91
18 and 20 X 16 oz.- Rolls	42.21	24.50	17.71

*C.R.=Cold-Rolled

Copper in Rolls - Miscellaneous

Consumer price - Item basis - 30,000 lb. or more.

Width, inches	Gage, inch	Price per pound, cents		Metal value, cents	Differential or spread, cents
		Rolls	Mill lengths		
Inc. ½ to 2	0.0508 & heavier	38.41	40.21	24.50	13.91 and 15.71
Inc. 2 inc. 8	0.0508 & heavier	37.21	37.81	24.50	12.71 and 13.31
Over 8 inc. 12	0.0508 & heavier	37.81	38.41	24.50	13.31 and 13.91
Over 16 inc. 20	0.0508 & heavier	39.01	39.61	24.50	14.51 and 15.11
Inc. ½ to 2	.032	38.41	40.21	24.50	13.91 and 15.71
Inc. 2 inc. 8	.032	37.81	38.41	24.50	13.31 and 13.91
Over 8 inc. 12	.032	38.41	39.01	24.50	13.91 and 14.51
Over 16 inc. 20	.032	40.21	40.81	24.50	15.71 and 16.31
Inc. ½ to 2	.0201	39.61	42.01	24.50	15.11 and 17.51
Inc. 2 inc. 8	.0201	38.41	39.01	24.50	13.91 and 14.51
Over 8 inc. 12	.0201	39.61	40.21	24.50	15.11 and 15.71
Over 16 inc. 20	.0201	43.21	43.81	24.50	18.71 and 19.31
Inc. ½ to 2	.010	41.41	45.01	24.50	16.91 and 20.51
Inc. 2 inc. 8	.010	40.21	42.01	24.50	15.71 and 17.51
Over 8 inc. 12	.010	40.81	42.61	24.50	16.31 and 18.11
Over 16 inc. 20	.010	50.41	52.21	24.50	25.91 and 27.71
Inc. 2 inc. 8	.005	43.81	48.61	24.50	19.31 and 24.11

Sheet Yellow Brass - Miscellaneous

Consumer price, 30,000 lb. or more - Item basis.

Width, inches	Gage, inch	Price per pound, cents		Metal value, cents	Differential or spread, cents
		Coils	Mill lengths		
Inc. ½ to 2	0.0508 & heavier	37.48	38.68	22.49	14.99 to 16.19
Inc. 2 inc. 8	0.0508 & heavier	36.28	36.88	22.49	13.79 to 14.39
Over 8 inc. 12	0.0508 & heavier	37.48	38.08	22.49	14.99 to 15.59
Over 12 inc. 16	0.0508 & heavier	38.68	39.28	22.49	16.19 to 16.69
Over 16 inc. 20	0.0508 & heavier	42.28	42.28	22.49	19.79 to 20.39
Over 20 inc. 24	0.0508 & heavier	48.28	48.88	22.49	25.79 to 26.39
Inc. ½ to 2	.032	37.48	38.68	22.49	14.99 to 16.19
Inc 2 inc. 8	.032	36.88	37.48	22.49	14.39 to 14.99
Over 8 inc. 12	.032	37.48	38.08	22.49	14.99 to 15.59
Over 12 inc. 16	.032	39.88	40.48	22.49	17.39 to 17.99
Over 16 inc. 20	.032	42.28	42.88	22.49	19.79 to 20.39
Over 20 inc. 24	.032	48.28	48.88	22.49	25.79 to 26.39
Inc. ½ to 2	.0201	38.68	40.48	22.49	16.19 to 17.99
Inc. 2 inc. 8	.0201	27.48	38.08	22.49	14.99 to 15.59
Over 8 inc. 12	.0201	38.68	39.28	22.49	16.19 to 16.79
Over 12 inc. 16	.0201	41.08	41.68	22.49	18.59 to 19.19
Over 16 inc. 20	.0201	44.68	45.28	22.49	22.19 to 22.79
Inc. ½ to 2	.0159	39.28	41.08	22.49	16.79 to 18.59
Inc. 2 inc. 8	.0159	38.08	39.88	22.49	15.59 to 17.39
Over 8 inc. 12	.0159	39.28	41.08	22.49	16.79 to 18.59
Inc. ½ to 2	.010	40.48	42.28	22.49	17.99 to 19.79
Inc. 2 inc. 8	.010	39.28	41.08	22.49	16.79 to 18.59
Over 8 inc. 12	.010	40.48	42.28	22.49	17.99 to 19.79
Inc. ½ to 2	.005	43.48	47.08	22.49	20.99 to 24.59
Inc. 2 inc. 8	.005	42.88	45.88	22.49	20.39 to 23.39
Over 8 inc. 12	.005	47.08	50.08	22.49	24.59 to 27.59

Naval Brass Condenser-Tube Sheets (Plates)

Consumer prices - Item basis - 30,000 lb. or more.

Size, inches	Selling price per lb., cents	Metal Value, cents	Differential or spread, cents
60 x 14 3/4 x 5/16 ½ hard	52.07	22.78	29.29
40 dia. x 1 1/8 thick H.R.*	45.07	22.78	22.29
87½ x 195 x 1¼ H.R.(½ circle)	56.07	22.78	33.29
100 x 139 x 1½ H.R. (Pattern Sheet - 2 or more curved cuts)	56.07	22.78	33.79

*H.R. = Hot-Rolled

Round Wire in Commercial Coils

Consumer price - Item basis - 30,000 lb. or more.

Size, inches	Selling price per lb., cents	Metal value, cents	Differential or spread, cents
No. 4 (.2043) Yellow Brass	36.57	22.49	14.08
No. 8 (.1285) Yellow Brass	37.17	22.49	14.68
No. 10 (.1019) Yellow Brass	37.77	22.49	15.28
No. 12 (.0808) Yellow Brass	39.57	22.49	17.08
No. 16 (.0508) Yellow Brass	41.97	22.49	19.48
No. 20 (.0320) Yellow Brass	44.97	22.49	22.48
No. 25 (.0179) Yellow Brass	48.57	22.49	26.08
No. 30 (.0100) Yellow Brass	54.57	22.49	32.08
No. 4 18% n/s*	56.09	27.24	28.85
No. 8 18% n/s	57.89	27.24	30.65
No. 10 18% n/s	59.69	27.24	32.45
No. 12 18% n/s	60.89	27.24	33.65
No. 16 18% n/s	63.29	27.24	36.05
No. 20 18% n/s	70.49	27.24	43.25
No. 4 Grade A 5%	58.67	28.87	29.80
No. 8 Grade A 5%	60.47	28.87	31.60
No. 10 Grade A 5%	62.27	28.87	33.40
No. 12 Grade A 5%	63.47	28.87	34.60
No. 16 Grade A 5%	65.87	28.87	37.00
No. 20 Grade A 5%	73.07	28.87	44.20

*n/s = Nickel Silver

18-percent Nickel Silver Rods in Mill Lengths

Consumer prices - Item basis - 30,000 lb. or more.

Size, inches	Selling price per pound, cents	Value of metal, cents per pound	Differential or spread, cents
3/8 round	70.49	27.24	43.25
5/8 round	68.09	27.24	40.85
$1\frac{1}{2}$ round	62.09	27.24	34.85

5-percent Phosphor or Bronze Rod in Mill Lengths

Size, inches	Selling price per pound, cents	Value of metal, cents per pound	Differential or spread, cents
3/8 round	60.47	28.87	31.60
5/8 round	58.67	28.87	29.80
$1\frac{1}{2}$ round	61.67	28.87	32.80

Copper Rods in Mill Lengths

Consumer prices - 30,000-lb. items or more.

Size, inches	Selling price per pound, cents	Value of metal, cents per pound	Differential or spread, cents
3/8 round	38.68	24.50	14.18
5/8 round	36.28	24.50	11.78
2 3/4 round	39.88	24.50	15.38
$\frac{1}{4}$ by 4 rect. (Bus bar)	37.48	24.50	12.98
$\frac{1}{4}$ square	39.88	24.50	15.38
5/8 square	38.68	24.50	14.18
2 square	37.48	24.50	12.98

Yellow Brass Free-Cutting Rods in Mill Lengths

Consumer prices - 20,000 lb. or more - Round, hexagonal, and square - Order basis - Other shapes, item basis.

Size, inches	Selling price per pound, cents	Value of metal, cents per pound	Differential or spread, cents
3/8 round	31.90	22.13	9.77
3/8 hexagon	33.10	22.13	10.97
3/8 square	34.30	22.13	12.17
3/8 by 5/8 rectangle	36.70	22.13	14.57
5/8 round	30.70	22.13	8.57
5/8 hexagon	31.90	22.13	9.77
5/8 square	33.10	22.13	10.97
5/8 by 2 rectangle	35.50	22.13	13.37
$2\frac{1}{2}$ round	31.90	22.13	9.77
$2\frac{1}{2}$ hexagon	33.10	22.13	10.97
$2\frac{1}{2}$ square	34.30	22.13	12.17
Rod for forging-rounds, all sizes	30.20	22.13	8.07
Rod for forging-other shapes	32.60	22.13	10.47

18-percent Nickel Silver Rods in Mill Lengths

Consumer prices - Item basis - 30,000 lb. or more.

Size, inches	Selling price per pound, cents	Value of metal, cents per pound	Differential or spread, cents
3/8 round	70.49	27.24	43.25
5/8 round	68.09	27.24	40.85
1½ round	62.09	27.24	34.85

5-percent Phosphor or Bronze Rod in Mill Lengths

Size, inches	Selling price per pound, cents	Value of metal, cents per pound	Differential or spread, cents
3/8 round	60.47	28.87	31.60
5/8 round	58.67	28.87	29.80
1½ round	61.67	28.87	32.80

Copper Rods in Mill Lengths

Consumer prices - 30,000-lb. items or more.

Size, inches	Selling price per pound, cents	Value of metal, cents per pound	Differential or spread, cents
3/8 round	38.68	24.50	14.18
5/8 round	36.28	24.50	11.78
2 3/4 round	39.88	24.50	15.38
$\frac{1}{4}$ by 4 rect. (Bus bar)	37.48	24.50	12.98
$\frac{1}{4}$ square	39.88	24.50	15.38
5/8 square	38.68	24.50	14.18
2 square	37.48	24.50	12.98

Yellow Brass Free-Cutting Rods in Mill Lengths

Consumer prices - 20,000 lb. or more - Round, hexagonal, and square - Order basis - Other shapes, item basis.

Size, inches	Selling price per pound, cents	Value of metal, cents per pound	Differential or spread, cents
3/8 round	31.90	22.13	9.77
3/8 hexagon	33.10	22.13	10.97
3/8 square	34.30	22.13	12.17
3/8 by 5/8 rectangle	36.70	22.13	14.57
5/8 round	30.70	22.13	8.57
5/8 hexagon	31.90	22.13	9.77
5/8 square	33.10	22.13	10.97
5/8 by 2 rectangle	35.50	22.13	13.37
$2\frac{1}{2}$ round	31.90	22.13	9.77
$2\frac{1}{2}$ hexagon	33.10	22.13	10.97
$2\frac{1}{2}$ square	34.30	22.13	12.17
Rod for forging-rounds, all sizes	30.20	22.13	8.07
Rod for forging-other shapes	32.60	22.13	10.47

Yellow Brass Tubes in Mill Lengths

Consumer price - Item basis - 10,000 lb. or more.

O. D., inches	Gage, inch	Selling price per pound, cents	Metal value, cents	Differential or spread, cents
$1\frac{1}{2}$ to 2	No. 6(.203)	41.09	22.49	18.60
$1\frac{1}{2}$ to 2	No. 7(.180) to 12(.109), inc.	39.79	22.49	17.30
2 to $2\frac{1}{4}$	No. 6(.203)	41.09	22.49	18.60
2 to $2\frac{1}{4}$	No. 7(.180) to 12(.109), inc.	39.79	22.49	17.30
$2\frac{1}{4}$ to $2\frac{1}{2}$	No. 6(.203) to 12(.109), inc.	39.79	22.49	17.30
$2\frac{1}{2}$ to $3\frac{1}{4}$	No. 6(.203) to 11(.120), inc.	39.79	22.49	17.30
$3\frac{1}{4}$ to 4	No. 6(.203) to 9(.148), inc.	39.79	22.49	17.30

Copper Tubes in Mill Lengths

Consumer price - Item basis - 30,000 lb. or more.

O. D., inches	Gage, inch	Selling price per pound, cents	Metal value, cents	Differential or spread, cents
$1\frac{1}{2}$ to 2	No. 6(.203)	40.92	24.50	16.42
$1\frac{1}{2}$ to 2	No. 7(.180) to 12(.109), inc.	39.72	24.50	15.22
2 to $2\frac{1}{4}$	No. 6(.203)	40.92	24.50	16.42
2 to $2\frac{1}{4}$	No. 7(.180) to 12(.109), inc.	39.72	24.50	15.22
$2\frac{1}{4}$ to $2\frac{1}{2}$	No. 6(.203) to 12(.109), inc.	39.72	24.50	15.22
$2\frac{1}{2}$ to $3\frac{1}{4}$	No. 6(.203) to 11(.120), inc.	39.72	24.50	15.22
$3\frac{1}{4}$ to 4	No. 6(.203) to 9(.148), inc.	39.72	24.50	15.22

Commutator Copper Bars - Mill Lengths

Consumer price - Item basis - 30,000 lb. or more.

Widths, 3/4 inch to include 1 1/2 inch	Selling price per pound, cents	Value of metal, cents per pound	Differential or spread, cents
Thick edge 0.200 inch and over			
Thin edge .100 inch and over	37.03	24.50	12.53
Thin edge .050 inch to .100 inch	38.23	24.50	13.73
Thin edge .025 inch to .050 inch	38.23	24.50	13.73
Thick edge .150 inch to .200 inch			
Thin edge .100 inch to .200 inch	38.23	24.50	13.93
Thin edge .060 inch to .100 inch	38.83	24.50	14.33
Thin edge .045 to .060 inch	39.43	24.50	14.93
Thin edge .025 inch to .045 inch	40.03	24.50	15.53
Thick edge .100 inch to .150 inch			
Thin edge .100 inch to .150 inch	38.23	24.50	13.73
Thin edge .060 inch to .100 inch	39.43	24.50	14.93
Thin edge .045 inch to .060 inch	40.63	24.50	16.13
Thin edge .025 inch to .045 inch	43.03	24.50	18.53
Thick edge under .100 inch			
Thin edge .060 inch to .100 inch	40.63	24.50	16.13
Thin edge .045 inch to .060 inch	41.83	24.50	17.33
Thin edge .025 inch to .045 inch	45.43	24.50	20.93

BIBLIOGRAPHY

1. American Cyanamid Company, Heavy-Media Separation Process, Ore-Dressing Notes: No. 11, July 1942, 27 pp.

2. American Society for Metals, Metals Handbook: 1948 edition, 1,444 pp.

3. Bray, John L., Nonferrous Production Metallurgy: John Wiley & Sons, 2nd ed., 1947, p. 167.

4. Bureau of Mines, Information Circulars 7560, 6691.

5. Cody, B. H., Milling Practices at the Concentrator of Morenci, Reduction Works: Mining Technol., Tech. Paper 2195, vol. 2, July 1947, p. 4.

6. Counselman, T. B., "Fluosolids for Roasting": Eng. and Min. Jour., vol. 151, No. 3, March 1950.

7. Gardner, E. D., and Mosier, McHenry, Open-Cut Metal Mining: Bureau of Mines Bull. 433, 1941, 176 pp.

8. Hayword, Carle R., An Outline of Metallurgical Practice: D. Van Nostrand Co., New York, 2 nd ed., 1940, 690 pp.

9. Hyer, John W., Jr., Heavy-Density-Separation - a Review of its Literature: Colorado Sch. Mines Quart., vol. 43, No. 1, January 1948, 94 pp.

10. Jackson, Charles F., and Hedges, J. H., Metal-Mining Practice: Bureau of Mines Bull. 419, 1939, 512 pp.

11. Jackson, Charles F., and Hedges, J. H., Metal-Mining Practice: Bureau of Mines Bull. 419, 1939, pp. 356-384.

12. Jackson, Charles F., and Hedges, J. H., Metal-Mining Practice: Bureau of Mines Bull. 419, 1939, p. 239, from Maclennan, F. W. Trans. Am. Inst. Min. and Met. Eng., vol. 91, 1930, p. 3986.

13. Lidell, Donald M., Handbook of Nonferrous Metallurgy: McGraw-Hill Book Co., Inc., New York and London, 2nd ed., 1945, 721 pp.

14. Mathewson, Champion H., Modern Uses of Nonferrous Metals: Am. Inst. Min. and Met. Eng., New York, 1st ed., 1935, 427 pp.

15. Michell, F. B., The Practice of Mineral Dressing: Mine and Quarry Eng., Electrical Press, London, 1950, 391 pp.

16. Newton, Joseph, and Wilson, Curtis L., Metallurgy of Copper: Chapman & Hall, Ltd., London, 1942, 518 pp.

17. Richards, Robert H., Locke, Charles E., Schumann, Reinhardt, Textbook of Ore Dressing: McGraw-Hill Book Co., Inc., New York, 3rd ed., 1940, pp. 1-5.

18. Stoughton, Bradley, and Butts, Allison, Eng. Met.: McGraw-Hill Book Co., Inc., New York, 3rd ed., 1938, 525 pp.

19. The Dow Chemical Co., Flotation Fundamentals: San Francisco, 54 pp.

20. Wilkins, Richard A., and Bunn, E. S., Copper and Copper Base Alloys: McGraw-Hill Book co., Inc., New York, 1st ed., 1943, 355 pp.

LIST OF COMPANIES AND NAMES OF PERSONS THAT FURNISHED INFORMATION, PHOTOGRAPHS, AND DRAWINGS

Farrell-Birmingham Co., Ansonia, Conn.	G. E. Schaefer	Strip and roll rolling
Lewis Foundry & Machine, Pittsburg, Pa.	Wm. B. Hachett	Rolling
Continental Industrial Engineers, Inc., 176 W. Adams St., Chicago, Ill.	Wm. Darrah	Annealing
The Electric Furnace Co., Salem, Ohio	C. H. West	Do.
Waterbury Farrell Foundry & Machine Co., Waterbury, Conn.	F. S. Van Valkenburg I. H. Talles	Wire-drawing machinery
Aetna-Standard Engineering Co., Youngstown, Ohio	H. G. Coffey	Wire-drawing Seamless tubing
General Cable Corp., Perth Amboy, N.J.	O. Garner	Flow chart, wire fabrication
Torrington Mfg. Co., Torrington, Conn.	R. S. Storrs	Mold equipment Slitters and milling machine
Wolverine Tube Co., Detroit, Mich.	H. A. Harty	Flow chart seamless tubing
	W. G. Schuryler 250 W. 57th St. New York, N.Y.	Continuous rod machine
Dings Magnetic Separator Co., Milwaukee, Wis.	W. B. Porter	Scrap separator
American Brass Co., Waterbury, Conn.	E. M. Pendleton	Prices scrap
Chase Brass & Copper Co. 1121 E. 260th St., Cleveland, Ohio	R. Ely Wm. F. Aylard	Scrap

Copper & Brass Research Association, 420 Lexington Ave., New York, N.Y.	T. E. Veltford	List of fabricators

III. RESOURCES

A. SUMMARY AND CONCLUSIONS

For the present economy, the world is plentifully endowed with copper deposits, and the mines are well-equipped to fill normal peacetime requirements. The resources, however, are not developed sufficiently to fill abnormal needs quickly. Extra plant capacity is lacking. More copper for emergencies is not a problem of finding ore; rather it is a matter of imports, stock-piles, new mine-plant facilities, and buying scrap.

At the same time, it should be remembered that large, war-induced expansions of plant and mine capacity might be uneconomic under normal peacetime marketing conditions. Consideration of markets has led to a delicate international economic balance in the present development of world copper resources.

A change of a few cents per pound in the price of copper makes a great difference in the tonnage of reserves that are profitable and unprofitable at any given time. Nevertheless, the actual physical resources are tremendous. To emphasize the point, it is possible to say that the known copper resources in the ground are adequate indefinitely for any foreseeable demands of peace or war. However, excess production would be costly, and 3 to 5 years are needed for mine development and new plant construction.

FIGURE III - I
COPPER RESOURCES

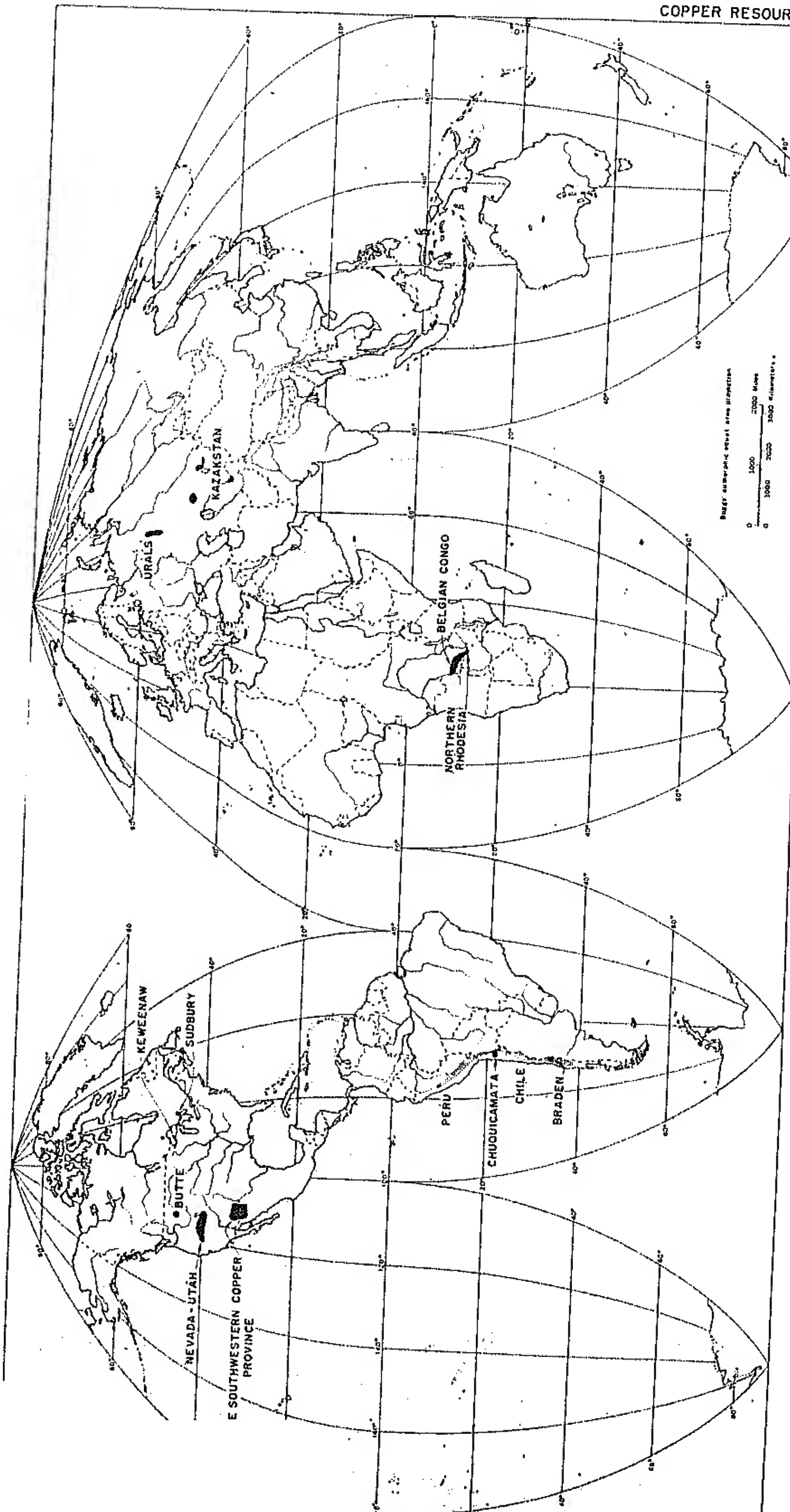


Figure III - I The location of 95 percent of the copper resources of the world, 1950

B. MAJOR RESOURCES

Copper is so widespread in nature that almost every country has some copper ore deposits; 36 nations produced it in 1950, and a few others may have submarginal or undiscovered deposits; yet the majority of the world's copper is concentrated in a few places. The scrap supply is mostly in the industrial areas, and about 90 percent of the world's unmined copper resources is concentrated in four regions: (1) South-central Africa, (2) Chile, (3) the western United States, (4) Kazakhstan, Russia, in that order. In Canada the Sudbury district and southern Quebec may be a close rival of Kazakhstan as a copper region, but comprehensive data on reserves are lacking. Figure III-1 shows the location of 95 percent of the copper resources of the world.

None of these reserve areas is a major consuming region; hence, transportation is an important feature in the copper industry. Ocean transport is used for over three-fourths of the world's copper production. Internal transportation to seaports is now an impediment to exploitation of the African deposits. Russia's railroads probably are an even greater handicap. In this respect, the United States is one of the most-favored nations. Most United States copper moves from the west coast by steamer to the fabricating plants on the east coast. However, the United States rail system connects all its producers and consumers, and less than 100 miles of additional track is needed to reach all the major undeveloped deposits.

C. COPPER DEPOSITS

Probably no two mining people could agree on a listing of world copper deposits in order of magnitude. One reason is that some of the deposits are so huge that they have not been fully explored. Many known reserves are so large they cannot be mined out in 30 years. Estimating profitability that far in the future is a dubious undertaking for many mines, because outside economic changes will affect the margin of profit and hence the quantity of profitable ore.

Predicting economic conditions 10, 20, or 30 years in advance seems much more hazardous than the mathematical calculation of tonnage in the ground for such periods, hence the following table is a compromise between developed reserves that are surely economic and partly explored semieconomic deposits that are so large that they may be important for the future. The coverage is based on tabulations of world reserves compiled in January 1949, for the National Security Resources Board by the United States Department of the Interior.

This list emphasizes the concentration of most of the world's copper reserves in a few places. Also notable is omission of many mines whose names are famous but whose reserves are not known to contain copper in quantities greater than 3,000,000 tons of copper metal.

TWELVE DISTRICTS OR MINES CONTAINING 85 PERCENT
OF THE WORLD'S COPPER RESOURCES, 1950

Deposits	Country	Major ownerships	Nationality
1. "Mine Series"	Northern Rhodesia	Selection Trust, Ltd., and Anglo-American Corp., Ltd.	British
2. Chuquicamata	Chile	Anaconda Copper Mining Co.	American
3. "Mine Series"	Belgian	Union Miniére du Haut Katanga	Belgian
4. Butte, Mont.	U. S.	Anaconda Copper Mining Co.	American
5. Braden (El Teniente)	Chile	Kennecott Copper Corp.	Do.
6. Bingham, Utah	U. S.	Do.	Do.
7. Keweenaw, Mich. a/	Do.	Copper Range Co. and Calumet & Hecla Consolidated Copper Co.	Do.
8. Morenci, Ariz.	Do.	Phelps Dodge Corp.	Do.
9. Sudbury, Canada b/	Canada	International Nickel Co. of Canada, Ltd., and Falconbridge Nickel Mines, Ltd.	Canadian
10. San Manuel, Ariz. a/	U. S.	Magma Copper Co. (Newmont Mining Co.)	American
11. Kazakhstan	U.S.S.R.	U.S.S.R.	Russian
12. Urals region	Do.	Do.	Do.

a/ Large reserves are considered marginal awaiting production tests.
b/ Copper is a coproduct of nickel production.

Coverage is raised to 93 percent by the addition of 13 districts as follows:

a. United States - Ely-Kimberly, Ray, Chino, Ajo, Bisbee, Yerington, Miami-Inspiration.
b. Mexico - Cananea.
c. Chile - Potrerillos (Andes Copper Co.), Aguirre-Africana (Santiago), Rio Blanco.
d. Peru - Toquepala-Quelleveco, Cerro de Pasco.

The remaining 7 percent of world reserves is distributed among 34 countries, according to a list compiled in January 1949 for the National Security Resources Board by the Department of the Interior. They have been described in the two volumes entitled "Copper Resources of the World", prepared by a committee of the 16th International Geological Congress, Washington, 1935, and published by the George Banta Publishing Co., Menasha, Wis. This is the most-comprehensive, detailed compendium of geologic descriptions of copper deposits of the world and is not repeated in this brief outline.

The story of 12 great mines is well-documented in "The Porphyry Coppers", by A. B. Parsons, published by the American Institute of Mining and Metallurgical Engineers, New York, 1933. Geology, mining methods, and costs are detailed in "Copper Mining in North America", Bureau of Mines Bulletin 405, 1938.

223535 O—52——17

D. ECONOMICS OF RESERVES

Ultimate copper resources are an important aspect for long-range planning, but reserves are normally considered in terms of economically profitable material. The United States reserve position in the light of current profitability ranks first 1/, at present, among the three major copper regions of the world, but the United States may be a poor third in the long run. The main reason is the relatively lean quality of domestic ore.

The African deposits are notable for huge reserves of 3-percent to 6-percent ore, whereas the United States average is less than 1 percent. The Chilean deposits have huge tonnages averaging 2 percent copper. Russian grades appear to be no better than those of the United States.

In spite of their advantages in grade of ore, several factors have militated against profits and greater expansion of foreign deposits. Probably the most important factor in United States leadership in copper mining is confidence in political and social stability, which encourages the vast investments in mechanization, thus making profitable the working of very low grade material. Also of major importance is the fact that the United States tariff policy protects, or threatens to protect, the world's greatest market against undesired competition. The size of the European market has limited the expansion of the African and Chilean copper mines.

Currently Africa has transportation difficulties and a shortage of fuel, power, and skilled labor. In South America the labor problem is intensified by the vagaries of local government policies. The threat of extreme taxation and even confiscation has deterred investments in Latin America. Political events will be a major factor in determining the competitive position of the world's major copper producers.

A physical factor is that three-fourths of American production now comes from large-scale, low-cost, open-pit deposits in contrast to the underground mines in South America 2/ and Africa. However, American open-pit deposits face eventual termination because of

---

1/ On the basis of apparent costs of production cited by Malozemoff, P., The Real Danger of High Costs: Eng. and Min. Jour., vol. 150, January 1950, p. 72.
2/ The great exception is Chuquicamata, which will operate an open pit for many years more while underground operations develop.

increasingly adverse ratios of waste rock that must be removed to uncover copper ore. At least under current conditions, the life of American open-pit operations is limited, some will last 10 years, some 20 years, and those that persist another 30 years will be the exceptions. Some, however, may be able to change to underground mining.

Recent discoveries and major unequipped properties in the United States are low-grade deposits. On the basis of the physical characteristics of the deposits, it must be emphasized that most American deposits have a temporary present advantage but a distinct long-range disadvantage when compared with the richer and larger African and Chilean deposits.

F. HISTORICAL REVIEW OF WORLD RESOURCES

The history of copper has been marked by shifting dominance among countries and deposits. This has been true from antiquity; China, south central Africa, Cyprus, Spain, Germany, Japan, England, and Chile have all had their periods of leading the world, and the record would lead one to suspect that such changes will recur in the future.

In modern industrial times the changes have been rapid. The Keweenaw district of Michigan rose to prominence on the demands of the electrical industry for this extraordinarily pure copper. Nowhere had "native" copper constituted an important deposit until, in the last half of the nineteenth century, "Lake copper" dwarfed all the world's previous production. The Calumet & Hecla Co. mined 40 percent of it.

Even before the turn of the century, however, Michigan was surpassed by the more common vein-type of deposit in "the richest hill on earth" - Butte, Mont. - and the Anaconda Co. became a powerful factor in the copper industry.

In the early years of this century, the copper world underwent another revolution. Extraordinary conditions of mineralization and erosion in the West had created a type of huge, low-grade, disseminated-copper deposit which had baffled previous small-scale operators. By means of power shovels, railroad-car haulage, and huge concentrating mills, D. C. Jackling showed the world that, by mass production, the deposits could be worked profitably. The rush was on to bring 10 huge new copper mines of this type into production. These disseminated deposits (porphyry coppers) then dominated the world copper market. Consolidations account for the present prominence of the Phelps Dodge and Kennecott groups. The Anaconda Co. assured itself a prolonged life by buying the greatest deposit of all - Chuquicamata, in Chile.

Expanding industry was able to absorb the meteoric rise in copper production until the depression of the 1930's, but the periods of revolutionary development were not over. In spite of the depression, English and American financiers opened on a large scale another geologic oddity of fabulous proportions - the steeply inclined, layered-copper deposits of Northern Rhodesia. These, with their counterparts in adjacent Belgian Congo, probably contain more copper than any other region in the world. Financial control of the African copper lies in London and Belgium. During the 1930's, Russia conducted a very successful prospecting and development campaign.

The decade of the 1940's brought into production in the United States the Morenci, Bagdad, Castle Dome, and Cananea open-pit mines and furnished several major discoveries and reserve developments, notably at Yerington, Nev., and San Manuel, Ariz. The great quantities of layered sulfide ore in the White Pine area of the old Keweenaw district of Michigan can also be counted a new discovery of the 1940's, although exploratory drilling actually was begun in 1937 on the extension of World War I mining operations. Russia probably expanded production greatly in the last decade.

The processes of discovery and development are continuing in the 1950's. What they will do to the shifting field of copper production is in large part governed by politics and economics. One point to remember is that cartels or syndicates have not been able to control the production or the price of copper without eventual failure. (3)3/ Too many byproduct mines and marginal mines are ready to seize upon a price advance, and the supplies of scrap copper are even more effective in breaking monopoly attempts. Periods of high prices have been stimulants to exploration and development of new properties.

Copper is threatened by competition from aluminum, even in copper's stronghold, the electrical field. This situation has strategic implications, because the United States is relatively poor in good sources of aluminum. Foreign aluminum resources may be balanced by the ability of African and Chilean copper mines to supply the world with copper at a strongly competitive price.

In a historical review of the world copper resources, it is worthwhile drawing attention to an authority ( ) writing in 1937; his conclusions are still applicable. Even the present reserve figures are of the same order of magnitude:

> Present world reserves of unmined copper aggregate 92,000,000 tons of metal, outside of the U.S.S.R., with indications of substantial additions during the next decade. This should be ample for the needs of the present century.
>
> Production costs as a whole have steadily lessened over a long period of years, and intrinsic costs, adjusted to a constant purchasing currency, while not exhibiting the marked fluctuations of actual costs, nevertheless show the same trend.

3/ Numerals in parenthesis refer to items in the selected references section of the bibliography at the end of the chapter.

> While the actual price in terms of currency may exhibit a marked advance if war or severe inflation takes place, the long-term trend of intrinsic copper prices shows a steady decline in the relative value of the metal in terms of its ability to purchase other commodities. This suggests that the markets of the world are aware of the fact that regardless of temporary expedients to control production and prices, there will be many sources that will produce copper cheaply in ample volume for the world's needs.

Another authority on world copper, Arthur Notman (2), wrote in 1935:

> The estimated known reserves of metal in the ground are about 100,000,000 tons, more than twice the amount the world has consumed in the last 132 years. This total cannot be regarded as a possible or probable limit but merely records the current estimate of known reserves of metal contained in material from which it can be profitably extracted by present methods at normal prices. It is capable of practically indefinite expansion by the factors of technologic advance and higher price, as well as the discovery of new deposits.

Notman (5) also said:

> In considering the amount of known reserves of copper in the world, the most significant fact is that no important copper-producing area has yet been exhausted. The Spanish district Rio Tinto, known to have produced from the days of the Phoenicians, is still active. The Mansfeld area, in Germany, has a recorded production extending continuously from the 13th century. In Cornwall, though production has ceased, substantial amounts of copper are still known to exist, but the metal cannot be produced in competition with that from more favorable areas. The reduction in the average grade of ore that can be treated has played and will continue to play an important part in expanding reserves of recoverable copper in the world.

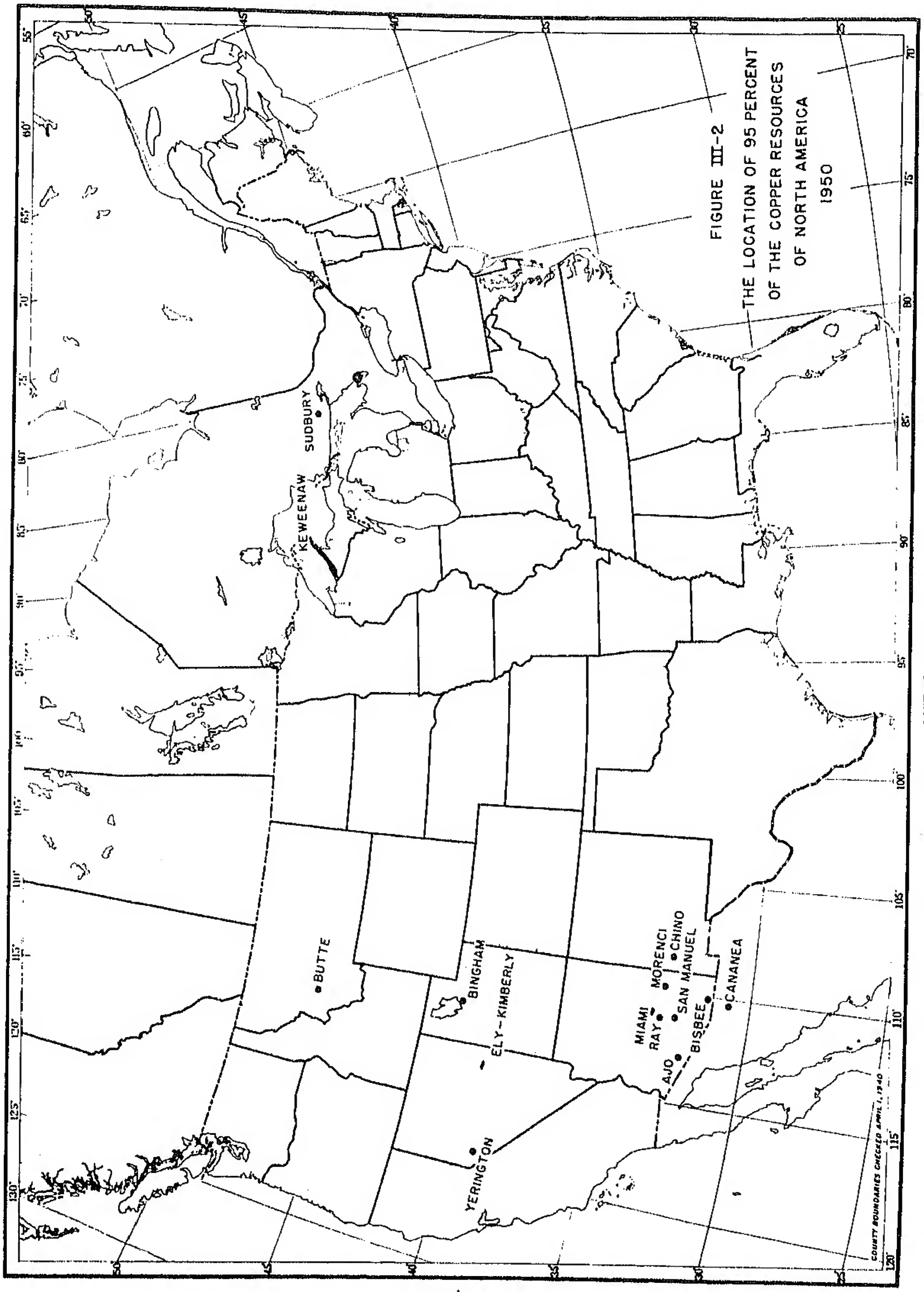


FIGURE III-2
THE LOCATION OF 95 PERCENT
OF THE COPPER RESOURCES
OF NORTH AMERICA
1950
SUDBURY
KEWEENAW
BUTTE
BINGHAM
ELY-KIMBERLY
YERINGTON
MIAMI
RAY
AJO
MORENCI
CHINO
SAN MANUEL
BISBEE
CANANEA
COUNTY BOUNDARIES CHECKED APRIL 1, 1940

F. UNITED STATES

1. Summary and Conclusions

The United States is in a healthy position with respect to the supply of copper known to remain in the ground. The present major producing mines have 20 to 30 years of life at the production rates of 1940-50. Beyond this, the exploration campaigns of the past decade around old mines and in new areas indicate an additional 10 to 15 years supply. If intensive exploration is continued, it is expected that this favorable position will be maintained. An important feature of the copper-reserve picture is that copper is a large-scale, mass-production resource. About 95 percent of the reserves occur in less than a dozen mines or mining areas.

The critical factor in utilizing copper resources under normal conditions is the highly competitive market. The economic conditions for profit and loss may change rapidly, but a tremendous quantity of copper is available in the ground. For each fraction of a percent that the grade of ore is lowered, large tonnages of copper are added to the supply.

2. Availability of Reserves

Much public attention is given to the importance of small mines, but it is a plain fact that the bulk of domestic copper comes from big operations; otherwise, the deposits that supply most of the output would be uneconomic mineral curiosities. The myriad little veins, lodes, and pods scattered throughout the country have national significance because some could be the yet-unrecognized clues to big deposits. They have relatively little importance as immediate producers of large tonnages of war-emergency copper, but they can have supreme importance for future discovery and long-term development, which should be made to keep pace with extraction.

Copper can be offered from hundreds of places; but, if the problem is to expand United States production surely and quickly, the job can be done with the greatest technical efficiency by concentrating on the few major deposits. This chapter does not deal with the engineering details of expanding the production; the subject here is simply the ore in the ground.

3. Distribution of Reserves

The United States has more large copper mines than any other country, but the accuracy of the knowledge of the reserves in individual deposits is extremely varied. For example, the San

Manuel deposit has been almost completely delimited, whereas the Bingham district is an unknown quantity, it is so large

Physical measurement of ore deposits is only one part of the problem of determining economic reserves. A greater difficulty is estimating the economic basis. Not only do the commercial reserve totals fluctuate with changes in the price of copper, but there is another feature that is often overlooked. Inflationary cycles can raise the cost of production and wipe out otherwise economic reserves.

Since 1930 American copper producers have been squeezed by a relative decrease in the purchasing power of copper compared with other commodities. Mining companies are acutely aware of the fluctuating profit-or-loss features of their reserves and tend to minimize them for safety's sake.

For these reasons, an outright statement of economic reserve figures is being avoided in this report; however, the many previous estimates of national reserves agree in a general way, and these data are used as a basis for coverage of the known reserves. Estimates for some of the large deposits have been revised in accordance with later information; figures for the small mines are taken from the Department of the Interior compilation of January 1949 for the National Security Resources Board. The deposits listed in the following table include both economic and semi-economic material. The resulting list is open to argument, because factual data are generally incomplete. Nevertheless, a preliminary attempt to show the concentration of reserves may be fruitful if it stimulates thought and further exploration. The omission of many famous mines from this list emphasizes the concentration of reserves in a relatively few places.

TWELVE MINING DISTRICTS CONTAINING 95 PERCENT
OF UNITED STATES COPPER RESERVES, 1950

District	Principal ownership
1. Butte, Mont.	1. Anaconda Copper Mining Co.
2. Bingham, Utah	2. Kennecott Copper Corp.
3. Keweenaw, Mich. a/	3. Copper Range Co. and Calumet & Hecla Consolidated Copper Co.
4. Morenci, Ariz.	4. Phelps Dodge Corp.
5. San Manuel, Ariz. a/	5. Magma Copper Co. (Newmont Mining Co.)
6. Ely-Kimberly, Nev.	6. Kennecott Copper Corp. and Consolidated Coppermines Corp.
7. Chino, N. Mex.	7. Kennecott Copper Corp.
8. Ray, Ariz.	8. Do.
9. Ajo, Ariz.	9. Phelps Dodge Corp.
10. Yerington, Nev. a/	10. Anaconda Copper Mining Co.
11. Miami, Ariz. b/	11. Miami Copper Co. and Inspiration Consolidated Copper Co.
12. Bisbee, Ariz.	12. Phelps Dodge Corp.

a/ Now being equipped for production.
b/ Includes Castle Dome, Copper Cities (Sleeping Beauty), Inspiration, Miami, and Globe.

Addition of the six following mines and districts brings the coverage to 98 percent: Magma, Ariz.; Silver Bell, Ariz.; Cornwall, Pa.; Bagdad, Ariz.; Tyrone, N. Mex.; and Glacier Peak, Wash. The remaining 2 percent of reserves is distributed among some 200 present and former copper mines and districts of the United States, most of which have been well-described in "Copper Resources of the World", volume 1, XVIth International Geological Congress. Only the economic features of the major deposits are described in this report.

Although the writer had access to many confidential reserve data and compiled initial work lists, no tabulations of reserves are given in this report because: (1) Many of the mining companies do not release reserve figures, (2) many of the published figures are incomplete, being only for proved and developed ore, (3) estimates of probable and possible reserves are subjective and generally based on incomplete information, (4) statistics of debatable accuracy draw attention away from the more important generalizations.

In the following descriptions of individual deposits, tonnage and grade figures have been given wherever they have been released and seem genuinely significant as to the magnitude of the deposit. Note, however, that the bases for the various figures are quite different; adding the figures would not yield an accurate total. In most cases the mine and district descriptions have been checked for accuracy by officials of the principal companies involved.

Keweenaw District, Mich.

Listing the Keweenaw district of Michigan among the nation's major copper reserves hinges on a very liberal definition of "reserves." Michigan's famous old native copper mines are considered only semieconomic deposits now, but they are estimated to contain roughly 200,000,000 tons averaging 1 percent copper above a cut-off grade of 0.5 percent copper. This is a grand total and includes inaccessible shaft pillars, stope pillars, and flooded areas that would be unprofitable to mine.

New developments by the Copper Range Co. suggest 500,000,000 tons of about 1 percent sulfide copper ore in measured, indicated, and inferred classes. These latter reserves have been explored only recently; the property is at present (1951) being equipped for production. Heretofore, this deposit had been considered submarginal, but in 1950 plans were drawn up to put it into large-scale production.

The combined figure of about 7,000,000 tons of contained copper in the Keweenaw district is one of the largest reserves in the United States and warrants emphasis as a future reserve, if not as a present producer. No byproducts are expectable from Michigan copper ore.

Both the native and the sulfide copper occur in layers far enough below the surface to require underground mining, some of it at great depths. The thickness of the ore bodies is less than the requirement for cheap, large-scale mining by the orthodox western methods. By adapting methods used on the iron ranges and other

ideas and using conveyor-belt haulage to the surface, low per-ton costs (at least as low as many producing areas) are anticipated. The region (White Pine) will have relatively low operating costs, but quite high overhead costs; the two combined will make the cost higher per pound than for some western porphyries but lower than for some others, lower than Butte, and much below that for the old Michigan range. Mining costs have risen to the point that the present trickle of production is no clue to the district's previous importance. It still ranks third in total production in the world, with nearly 5,000,000 tons of metal. This pre-eminence was attained in spite of the difficulty of prospecting a region covered by a blanket of glacial sand and gravel. Outcrops of bedrock are rare. The possibility of additional ore bodies being concealed under the glacial debris has led to geophysical testing, but present methods have been unsuccessful.

Butte, Mont.

Butte, Mont., is the second-largest vein-type copper deposit in the world 4/ and has the world's largest total copper production. Current annual production is relatively low, however, being only about 50,000 tons of copper.

The multitude of veins in the central square mile of the district have been mined out on an average to a depth of about a half mile. The workings on some of the veins have reached a depth of 4,000 feet, but exploration at that depth is limited. The significant thing from the resource point of view is that the tonnage and grade of ore on the average have not decreased appreciably with depth. Apparently bottoming of the mines will be an economic problem rather than a change in the metal content of the deposit.

Can the veins persist another half mile in depth, on the average? It is an impressive possibility; Butte has already yielded over $2,500,000,000 from copper, lead, zinc, gold, silver, and manganese, including the value of 6,800,000 tons of copper metal.

An important resource fact about Butte is that, in addition to the distinctive veins, there are immense quantities of metal in the myriad veinlets and cracks and in disseminated specks in the wall rocks of the major veins. In recent years it has been profitable to ship the old waste dumps of the Butte mines for extraction of this copper.

4/ The largest is actually Chuquicamata, Chile, despite the fact that it is commonly called a "porphyry copper."

Also in recent years it has been recognized that a certain portion of the district can be mined en masse, by block-caving large tonnages of rock that average a quarter of the grade necessary for individual veins to be economic. The reserve of ore in this type above the 3,400 level has been published as 130,000,000 tons of ore averaging 1.1 percent copper, from which about 1,300,000 tons of copper metal can be extracted. The Anaconda Copper Mining Co. is planning to spend $27,000,000 to put this Greater Butte project into production.

Twenty years ago this sort of operation was not even considered. What might be expected in another 20 years? Will engineers give serious thought to the fact that there are greater tonnages of rock that average 0.5 percent copper? The total copper content above this grade would be many millions of tons.

The drainage water alone from the Butte mines yields about 3,000 tons of copper annually, which is more and cheaper copper than the annual output of many countries. Even if ordinary mining methods ceased at Butte, this pumping out of the drainage water and precipitation of the copper could go on almost indefinitely.

Bingham District, Utah

The geologic basis for appraising the Bingham district is more complex than that at Butte, and much information is lacking. Lawsuits over extralateral mining rights have discouraged publication of geologic descriptions, and the threat of taxation of reserves in the ground has discouraged long-range exploratory campaigns. Nevertheless, some facts are available. The Bingham open pit exposes an ore body whose dimensions are nearly 1 mile in average diameter and about one-half mile in exposed vertical range. Such a quantity of mineralization should persist in depth if the general geologic conditions continue; presumably they do.

Furthermore, the Kennecott Copper Corp. recently built a $16,000,000 refinery nearby. Generally such plants are not built unless they can be expected to have a life of at least 20 years. In terms of recent annual production, this would suggest that a total of over 5,000,000 tons of copper remains in the deposit.

Probably much more copper exists, but it may be difficult to handle. The open pit has already transformed a mountainside into a hole in the ground, and the deeper it goes, the more surrounding waste rock must be removed from the side hills. The stripping ratio in 1950 was 4 tons of waste to 3 tons of ore. Many years hence, the problem of whether to continue as an open pit, to start an underground mine, or to shut down will become acute. The first

two can be settled by engineers' cost estimates, but the latter will be affected by general economics of the copper industry.

The Bingham district has produced over 5,000,000 tons of copper metal and much lead, zinc, gold, silver, and molybdenum. Bingham's production totals are second only to Butte among the world's copper districts. Current annual production is the world's largest, about a quarter of a million tons.

Byproducts are an important feature of Bingham copper production. It has rivaled Climax as the world's greatest producer of molybdenum in recent years; it is the second-largest gold producer in the United States, and the district as a whole is the third-largest source of lead and silver. The last two are not associated with the production of copper.

Ely-Kimberly (Robinson) District, Nev.

The Ely-Kimberly district is a mineralized belt about 8 miles in length and 1 mile in width. The copper mineralization is largely in the form of the original unaltered sulfide mineral - chalcopyrite. The distribution is suggestive of large tonnages of deep, low-grade ore, which, however, would be more costly to mine than the present open-pit and previous underground operations. In addition to copper, this district is a notable producer of byproduct gold. A small quantity of molybdenum is associated with the copper. Total production has been about 1,700,000 tons of copper metal, and annual production in the late 1940's was about 45,000 tons, with no end in sight. In 1950 stripping began for another, but much smaller, open-pit mine in the district.

Yerington, Nev.

Yerington is an old district that produced a little high-grade copper. Its importance now stems from exploration and development work by the Anaconda Copper Mining Co. in the last decade. Like San Manuel, this is a successful example of modern exploration - long-shot geologic interpretation backed by large amounts of risk capital. Published data indicate an assured reserve of at least 50,000,000 tons averaging about 1 percent copper. The district may contain more ore, but even the proved reserves have not been equipped for production owing to the economic disadvantages of the venture compared with other Anaconda properties. Steps are now being taken to bring it into production for defense needs.

Chino, Santa Rita District, N. Mex.

Chino is a compact, open-pit deposit with an appreciable content of molybdenite. Highly successful leaching of copper from the waste-dump rock is a notable feature of the chalcocite mineralization. Open-pit operations are limited toward the south and east because of adverse waste-stripping ratios. Total production has been about 1,400,000 tons of copper metal. Annual production is around 60,000 tons.

Miami District, Ariz.

The name "Miami district" is used here to cover a broad, 15-mile zone from Castle Dome to Globe and from Miami to Copper Cities (Sleeping Beauty). Mineralization is not continuous, but the quantity of copper found in the explored areas is remarkable. Much of the district is concealed by deep valley fills that have prevented prospecting and mining.

Published reserve figures credit the Inspiration mine with about 400,000 tons of copper remaining and Miami with about 100,000 tons. Miami is a deep underground mine but is making money on about the lowest-grade ore in the United States. The Inspiration mine recently converted largely to open-pit operations in the expectation of reducing costs on ore comparatively near the surface.

At the Globe end of the district, the Old Dominion mine yielded a large output of rich ore until the beginning of the depression in 1930. Some reserves are known to exist, but they are comparatively small.

Castle Dome has but little ore left. Copper Cities has been drilled thoroughly by the Miami Copper Co., and the Reconstruction Finance Corporation has recently approved a loan to develop the property and move the Castle Dome concentrator to this new location. The production planned from Copper Cities will equal that previously obtained from Castle Dome.

Morenci, Ariz.

The Morenci-Metcalf district of Arizona was famous as a high-grade copper producer; it has been resurrected as an even greater low-grade, large-scale producer. Reserve estimates made in the 1930's were, in round numbers, close to 300,000,000 tons of ore grading about 1 percent copper and, unofficially, 450,000,000 tons at 1.1 percent. The difference in the figures is less significant than the fact that general economic changes can make a greater difference in the quantity of profitable ore. About 1,000,000 tons

of copper have been mined from the open pit and a grand total of 1,900,000 tons of copper metal has been extracted from the district. The ratio of waste stripping to ore mined in this open pit was about 1.75:1 in 1950, and it is not expected to get much worse for many years.

The Phelps Dodge Corp. in 1942 completed a $45,000,000 plant; then the Reconstruction Finance Corporation spent $25,000,000 expanding to a concentrator capacity of more than 45,000 tons of ore daily and smelter output of 150,000 tons of copper, plus a steam power plant and costly water supply. An investment such as this by a farsighted company indicates a long-lived operation - least 20 to 30 years. At 150,000 tons annual copper production this would mean an ore body of the magnitude of 3,000,000 to 4,500,000 tons net yield of copper, recovery being about 85 percent. These figures bracket and check the ore-reserve estimates published early in the 1930's. Minor constituents of the ore are gold, silver, and molybdenum.

Ray, Mineral Creek District, Ariz.

Ray has recently developed an open-pit operation, in addition to its underground mine. The ore body is a result of enrichment of a copper deposit of very low grade by erosion and weathering. Thus, it can be expected to have comparatively sharp outlines and a definite cut-off at a relatively shallow depth. Fortunately it was a large deposit to begin with. Total production has been a little over 1,000,000 tons of copper metal and current annual production about 24,000 tons. Very minor constituents of the ore are gold, silver, and molybdenite.

San Manuel, Ariz.

The Magma Copper Co. owns, through its wholly owned subsidiary, San Manuel Corp., the major portion of the ore reserves at San Manuel. The entire deposit probably contains in round numbers, 500,000,000 tons of ore averaging nearly 0.8 percent copper and containing about 4,000,000 tons of the metal. About 1,000,000 tons of this copper is in the form of oxides, from which the recovery of copper might be only 80 to 85 percent, compared with 90 to 95 percent from the sulfide ore. Since the ore body is deeply buried and steeply inclined, underground mining will be required, but the large size of the ore body makes it amenable to low-cost, mass-production methods.

This deposit is not equipped for production as yet; it needs several years of underground and surface work, including construction

of a concentrator, railroad, townsite, and possibly a smelter; however, it has been thoroughly explored by drill holes from the surface, and underground development is well underway. The investment of sufficient funds to achieve large-scale production has caused some concern, considering the low grade of the ore. The content of gold and silver is low, but the molybdenum grade is relatively high for "porphyry-copper" deposits.

San Manuel is significant for more than its tonnage of copper. It highlights the trend to the modern type of discovery. The time-honored prospector had no part in its exploration; the mining engineer did not devise a new low-cost mining method; the metallurgist did not convert known low-grade rock into ore by developing new methods of metal recovery. This was fundamentally a case of correct geologic interpretation of geologic conditions which had defeated previous attempts at exploration. The Geological Survey recommended exploration in the spring of 1942, and the Bureau of Mines began drilling the fall of that year, as part of the War Minerals program.

Ajo, Ariz.

The New Cornelia copper ore body now consists largely of the original mineralization unaltered by weathering and erosion. Until 1930 only oxidized ore had been treated. The published geologic cross sections suggest rather definite, if as yet unascertained, boundaries to the mineralization, large as it may be. Total production has been about 1,200,000 tons of copper; annual production is about 60,000 tons. Valuable byproducts are gold and silver.

Bisbee, Ariz.

Bisbee is the fifth-largest copper district in the United States, judged by total past production, but its relative reserve position is doubtful. The ore has occurred chiefly as veins and replacement masses whose location and extent are difficult to predict. Unlike Butte, a depth ratio is not apparent.

The Phelps Dodge Corp. has kept a 5- to 7-year reserve developed at Bisbee, which means something between 100,000 and 200,000 tons of copper metal. However, this situation has persisted many years and presumably indicates a much larger quantity of undiscovered copper. How much of this will repay the risks and costs of exploration, development, and extraction is the critical problem.

One small "porphyry'copper" deposit was mined out at Sacramento Hill; another low-grade, open-pit, "porphyry-copper" deposit is designated the East Bisbee. This has not appeared to be economic

under peacetime conditions, but steps are being taken now to put it into production.

The relative costliness of underground mining conditions at Bisbee has curtailed the scale of operations to the point where the present production belies its past performance. On the record, it is a major copper area. In ore bodies adjacent to and sometimes mixed with copper are large proportions of lead, zinc, and silver, for which Bisbee is also famous.

G. CANADA

The Sudbury district, Ontario, has the largest known copper reserves in North America outside the United States. It is famous for huge, massive-sulfide ore bodies containing both copper and nickel in varying proportions but averaging about equal quantities. Company figures for proved ore alone total about 250,000,000 tons of 3-percent copper-nickel ore. This tonnage would contain about 3,750,000 tons of copper that can be classed as assured. In addition, the deposits contain unknown, but probably large, quantities of probable and possible ore.

Since nickel generally sells for two or three times the price of copper, the latter is in effect a byproduct that can be thought of as costing virtually nothing to produce. The International Nickel Co. can, in effect, profitably undersell all other large copper producers. Annual production is usually about one-seventh or one-eighth of the United States total. The platinum-group metals are valuable constituents of the copper-nickel ores.

All other Canadian copper mines yield a combined total that is slightly larger than that of the Sudbury district, but their reserves are not so well known. Important sources are the Rouyn-Noranda district in Quebec and the northern Saskatchewan-Manitoba region. A new development on the Gaspé Peninsula has been reported to consist of 57,000,000 tons of ore averaging 1 percent copper. Production at the rate of 5,000 tons of ore daily is planned by the owner, Noranda Mines, Ltd.

II. MEXICO

Cananea, in the State of Sonora, has a copper deposit that may be of the order of magnitude of the Ely, Chino, Ray, and Ajo group. It is 50 miles southwest of Bisbee, Ariz., in an area where all transportation, supplies, and the production of industrial consequence make the mine dependent on, and tributary to, the United States. Cananea is operated by the Cananea Consolidated Copper Co., which is owned by the Greene-Cananea Copper Co., which in turn is owned by the Anaconda Copper Mining Co., and is effect an American producer, except for such problems as labor unions, taxes, and tariffs.

During World War II Cananea exhausted the rich "pipe" ore bodies that made the district famous. Fortunately, the possibilities of large volumes of low-grade disseminated copper were correctly appraised. With the financial help of the Reconstruction Finance Corporation a new concentrator with a daily capacity of 16,000 tons of ore was built and an open pit begun. Excessive waste stripping has now curtailed operations in that open pit, but large-scale mining is proceeding underground, and another open-pit operation is contemplated. The ore is so low-grade that the mine probably cannot withstand adverse economic conditions.

The smelter at Cananea, with an annual capacity of 50,000 tons of copper, easily handles all the mill concentrates. Shipments are by rail via Naco, Ariz.

I. CHILE

1. Chuquicamata

Chuquicamata, 160 miles by rail northeast of the port of Antofagasta, is the world's largest economic copper deposit. It is owned by the Chile Exploration Co. which is controlled by the Chile Copper Co., a subsidiary of the Anaconda Copper Mining Co., New York. Since its beginning in 1916 the total production to date has been more than 5,000,000 tons of copper.

The ore zone measures a maximum of about 2 miles long and $\frac{1}{2}$ mile wide, with depth unknown. The copper minerals occur in close-spaced veins and veinlets; disseminated copper is not mentioned, (6) but the deposit is commonly classed as a "porphyry copper". Operations have been in the form of open-pit mining, followed by crushing, leaching and electrolysis. Past production has come only from oxide ores. The altitude of the mine is 9,300 feet, and the region is extremely arid.

Unofficial estimates dated 1935 (8) listed reserves as 938,931,000 tons of ore averaging 2.15 percent copper, with a content of 20,187,023 tons of copper. Current and planned rates of production are for about 270,000 tons of refined copper per year, indicating a life-expectancy of nearly a century. Currently the mining grade is down to 1.6 percent copper, because that is the grade of the available oxide ore in the open pit. This type of ore is gradually approaching exhaustion. To handle the underlying sulfide ore the Company is spending $90,000,000 to build a concentrating mill and smelter. An underground caving operation is also to be expected at some future date.

2. Braden

Another copper deposit so large its ultimate size is unknown is the Braden mine, or El Mineral Teniente, at Sewell, 140 miles by rail from the seaport at San Antonio. It is operated by the Braden Copper Co., owned by the Kennecott Copper Corp., New York.

Since modern operations were begun in 1912, Braden has produced 3,572,500 tons of copper; the annual capacity is currently about 150,000 tons. Reserves are commonly regarded as of the order of magnitude of 200,000,000 tons, averaging 2 percent copper. This figure has been popular (2) (1) for 20 years and may continue so for a much-longer period.

The Braden deposit is unique among the world's first-order copper producers for being located in and around an old explosive volcanic vent. Only underground mining has been practiced, and only sulfide

ore has been treated. Molybdenum is recovered. Operations are between altitudes of 7,400 and 9,600 feet.

3. Other Mines

Chile has another producing "porphyry-copper" mine at Potrerillos 5/ and many sizable deposits in the explorational stage. These are not known to rank now among the world's leaders, but their existence emphasizes the fact that one of the world's major copper provinces occurs in the northern half of Chile and extends into Peru.

5/ 94 miles by railroad from the seaport of Barquito. It is owned by the Andes Copper Mining Co., a subsidiary of Anaconda Copper Mining Co.

J. BELGIAN CONGO AND NORTHERN RHODESIA

Although the ownerships are quite different, the physical characteristics justify joint treatment of the copper deposits on the borders of these two countries. Copper occurs elsewhere in these countries, but it is the 300-by 50-mile belt of copper-bearing sedimentary rocks on the Congo-Rhodesian border that comprises the world's greatest copper province.

The copper occurs in oxide minerals in the upper few hundred feet of the deposits, elsewhere as sulfides disseminated through the rock. The ore bodies are remarkably confined to individual beds, chiefly sandstone, which dip steeply. The ore bodies are often scores of feet thick, so relatively large-scale underground mining is possible.

Reserves of the four principal Northern Rhodesian mines published in the 1949 annual reports of the various companies follow in rounded figures:

Mines	Crude ore, tons	Grade (% copper)	Copper content, tons
N'Changa	140,000,000	4.66	6,500,000
Mufulira	130,000,000	3.85	5,000,000
Rhokana	110,000,000	3.35	3,600,000
Roan Antelope	90,000,000	3.25	3,000,000
	470,000,000	3.85	18,100,000

These figures represent proved ore. The ultimate size of the deposits is unknown, but probably they contain many times as much copper as has been developed and reported to date.

Data on the reserves of the Belgian Congo mines are lacking. All the copper mines are owned by the Union Miniere du Haut Katanga. No authoritative estimates are known, but a figure of 10,000,000 tons of copper is widely quoted and seems conservative. Considering the occurrence, possibilities are good for reserves several times as great. Most of the 300-mile copper belt and most of the mines are in the Congo, but operational difficulties and production agreements have limited their development. The grade of ore is reported as 6-percent copper, which is the world's highest for large deposits.

This district is one of the most adversely situated of all major copper regions with respect to transportation, being nearly 1,500 miles by rail from either coast of Africa. Supplies of power, fuel, and railroad equipment, as well as labor, have been inadequate.

The annual copper production of the Belgian Congo has reached 180,000 short tons, and Northern Rhodesia's output was well over 300,000 short tons in 1950. In addition, expansion is planned or underway. Cobalt is an important byproduct of some of the mines; notably absent are gold, silver, molybdenum, and pyrite.

K. U. S. S. R.

Recent data are lacking. What appears to be the most comprehensive and precise available Russian report (4) on copper resources gives a total of 15,800,000 tons of copper content in deposits in the U.S.S.R. at the end of the first 5-year plan of exploration (1932). Exactly half the copper was reported to be in deposits of the "porphyry-copper" type, famous in the western United States. An excellent evaluation of the basic data has been made by Riddell and Jermain (7) in 1934. They report a total of 16,000,000 tons of copper, which checks the Russian figure but differs in detail. The grade of ore is 1.1 percent copper, approximately equal to the United States average.

By 1950 the reserve totals may well be larger. The reason is that only 5 years of scientific prospecting had been done before the foregoing estimates. Now there have been 23 years of prospecting by modern exploration techniques under one of the most massive prospecting campaigns in the world's history. It may be compared to the explorations of the West from 1849 to 1915 under a central administration.

The industrial region on both sides of the Urals benefits from the fact that numerous copper-pyrite deposits are scattered for several hundred miles through the southeastern Ural Mountains. The reserve of 200,000,000 metric tons of 2-percent copper ore quoted by Nekrasoff has been reported by Riddell and Jermain as 240,000,000 metric tons at 1.6 percent copper. The copper content is about 4,000,000 short tons in either case. An important feature to note is that the pyrite content of these ores is probably 30 to 40 times the copper content. Hence, large quantities are available for the manufacture of sulfuric acid, a product essential to industry. The crude ores also carry about $1 per ton in gold and silver, plus 1 to 3 percent zinc. Accurate data on their present annual production are not available.

The Russian "porphyry-copper" type deposits are described in terms that would be typical of those in the western United States, including the molybdenum content. Nekrasoff reports 7,900,000 tons of copper content. The Russian deposits apparently are much more numerous but not individually so large as those in the United States. A little over 2,000,000 tons of copper in a single deposit seems to be the upper limit. Kounrad on Lake Balkash and Almalyk to the southwest are the largest "porphyry coppers."

Many other sizable and productive copper deposits are described by Nekrasoff, notably the Rhodesian-type sandstone deposit at

Djezkazgan northwest of Lake Balkash. The inference is that in Kazakhstan, in a semicircle 400 miles from Lake Balkash, Russia has one of the world's great copper provinces, possibly comparable to that of the western United States. The deposits there, as in the western United States, are 500 to 2,000 miles from consuming centers, but rail transportation is thought to be a weakness in Russia.

The Bureau of Mines estimates Russia's total current annual production at 180,000 tons. Another source estimated for 1949 a total of 256,000 metric tons, which is roughly one-third of the United States production in that year. In appraising the Russian situation, it is probably still appropriate to quote Riddell and Jermain, writing in 1934 (7):

> The physical characteristics and location of the copper mines of the Soviet Union in respect to supplies and market will always continue a heavy handicap in world competition. (However) In all appraisals of industrial progress in U.S.S.R. one may well keep an open mind. The remote desolate wastes of Central Asia - where lie two-thirds of the listed Soviet copper reserves - are, after all, not much more inaccessible than were, not long ago, the resources of Central Africa, now offering supplies not unknown to copper circles on other continents. The slow tempo of current Soviet copper production is something of a menace to the industrial base of the nation and distinctly a challenge to the Kremlin. One is safe in taking the view that strong forces are being organized for the speeding up of this particular corner of the Soviet industrial picture and that this copper effort must and will be directed squarely at the Kazakhstan and Central Asian "porphyries."

BIBLIOGRAPHY

1. Barbour, P. E., World Copper-Ore Reserves: Eng. and Min. Jour., vol. 131, 1930, p. 178.

2. Croston, John J., Trends in Copper Production, Reserves, and Costs: A.I.M.E. Tech. Pub. 826, July 1937.

3. Federal Trade Commission, Report on the Copper Industry: United States Government Printing Office, Washington, D.C., Part I, 1947, p. 250.

4. Nekrasoff, B., Copper-Ore Regions of the U.S.S.R.: 16th Internat. Geol. Cong., Copper Resources of the World, Washington, D.C., vol. 1, 1935, pp. 649-662.

5. Notman, Arthur, Economic History of the Copper Industry: 16th Internat. Geol. Cong., Copper Resources of the World, Washington, D.C., vol. 1, 1935, p. 21.

6. Notman, Arthur, Economic History of the Copper Industry: 16th Internat. Geol. Cong., Copper Resources of the World, Washington, D.C., vol. 1, 1935, p. 31.

7. Riddell, Guy C., and Jermain, G. C., Russian Copper: Eng. and Min. Jour., vol. 135, 1934, pp. 547-551, and vol. 136, 1935, pp. 83-87.

8. Taylor, A. V., Jr., Ore Deposits of Chuquicamata, Chile: 16th Internat. Geol. Cong., Copper Resources of the World, Washington, D.C., vol. 2, 1935, pp. 473-484.